PROCEEDINGS

OF THE

LAKE SUPERIOR MINING INSTITUTE

TWENTY-FIRST ANNUAL MEETING
(Menominee Range)

Held at

BIRMINGHAM, ALABAMA

MARCH 13, 14, 15, 1917

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1917

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ISHPEMING, MICH.

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PAPERS.

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(Term one year).

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FRANK E. KEESE .... Ishpeming, Mich.
GRANT S. BARBER .... Bessemer, Mich.
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(Term expires 1917).

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(The above officers constitute the council).

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MURPHY, C. M............................... Ishpeming, Mich.
MURRAY, ROBERT .... Hibbing, Minn.
MYERS, ALBERT J...................... Iron Mountain, Mich.

M'CALLUM, JAMES G.......................... Houghton, Mich.
M'CARTHY, EDW. P..Minnesota School of Mines, Minneapolis, Minn.
M'CLURE, O. D............................... Ishpeming, Mich.
M'CORMICK, EDWARD .... Negaunee, Mich.
M'DONALD, D. B..................302 Glencoe Bldg., Duluth, Minn.
M'DONALD, L. N...................301 Glencoe Bldg., Duluth, Minn.
M'DOWELL, JOHN .... Hibbing, Minn.
M'GEE, M. B............................ Crystal Falls, Mich.
M'GONAGLE, WILLIAM A..............Wolvin Bldg., Duluth, Minn.
M'GREGOR, SILAS J...................... Iron Mountain, Mich.
M'INDOE, JAMES A............................ Norway, Mich.
M'INTYRE, JOHN E........................... Nogales, Arizona
M'KENNA, EDWARD B............Commercial Club, Duluth, Minn.
M'LEAN, JOHN H.............................. Duluth, Minn.
M'LEAN, RICHARD EARLE............... Wells, Delta Co., Mich.
M'NAMARA, THOMAS B......................... Ironwood, Mich.
M'NAIR, F. W............................... Houghton, Mich.

M'NEIL, E. D.. Virginia, Minn.
M'RANDLE, WILLIAM E. R........................ Bessemer, Mich.

NEELY, BENJAMIN C........................ Crystal Falls, Mich.
NELSON, JOHN E................................ Negaunee, Mich.
NEWBY, WILLIAM................Puritan P. O., N. Ironwood, Mich.
NEWETT, GEORGE A.......................... Ishpeming, Mich.
NEWETT, W. H................................. Ishpeming, Mich.
NEWTON, L. L...................... 68 E. Oak St., Chicago, Ills.
NICHOLAS, THOMAS J............................ Palmer, Mich.
NICHOLS, F. W.................................. Houghton, Mich.
NICKERSON, H. F............................... Houghton, Mich.
NIXON, JOHN A................925 Harris Trust Bldg., Chicago, Ills.
NOETZEL, BENJAMIN D...........123 Blanche St., Houghton, Mich.

OBERG, ANTON C...............503 Manhattan Bldg., Duluth, Minn.
ODGERS, IRA .... Crystal Falls, Mich.
OLCOTT, WILLIAM J.............................. Duluth, Minn.
OLSON, OSCAR E..............403 N. Lawrence St., Ironwood, Mich.
ORBISON, THOMAS W........................... Appleton, Wis.
ORR, FRANK D......................Lyceum Bldg., Duluth, Minn.
OSBORN, CHASE S...................... Sault Ste. Marie, Mich.
OSWALD, EUGENE H........................ Crystal Falls, Mich.
OVERPECK, HOLLIS W...................Box 617, Virginia, Minn.

PAINE, W. A........................82 Devonshire St., Boston, Mass.
PAINE, FRANCIS W............................ Houghton, Mich.
PARKER, RICHARD A............802 Equitable Bldg., Denver, Colo.
PASCOE, PETER W.............................. Republic, Mich.
PATRICK, RICHARD S.........314-15 Sellwood Block, Duluth, Minn.
PEARCE, E. L................................. Marquette, Mich.
PEARL, HOLMAN I..............P. O. Box No. 745, Rice Lake, Wis.
PELLENZ, WM. F. JR............Carson Lake, P. O., Hibbing, Minn.
PENGILLY, EDWARD .... Crystal Falls, Mich.
PENNIMAN, DWIGHT C............Curtis Court, Minneapolis, Minn.
PENTON, JOHN A...............Iron Trade Review, Cleveland, Ohio
PERKINS, SAMUEL J.............................. Diorite, Mich.
PERKINS, WILLIAM J............................... Alpha, Mich.
PETERSON, A. Y................................ Chisholm, Minn.
PITKIN, S. H.......................682 W. Market St., Akron, Ohio
POTTER, OCHA .... Houghton, Mich.
POTTER, W. T.................................. Ishpeming, Mich.
POWELL, D. W................................ Marquette, Mich.
POWELL, A. E................................ Marquette, Mich.
PRESCOTT, FRED M........................ Menominee, Mich.
PRESCOTT, L. L............................ Menominee, Mich.
PRESCOTT, C. S............................ Menominee, Mich.

PRYOR, R. C.. Houghton, Mich.
PURSELL, H. E.. Kewanee, Ills.
QUIGLEY, G. J.. Antigo, Wis.
QUINE, JOHN THOMAS..............413 Vine St., Ishpeming, Mich.
QUINN, CLEMENT KRUSE .... Virginia, Minn.
RAISKY, F. H.. Duluth, Minn.
RANKIN, WILLIAM A............................. Painesdale, Mich.
RAPP, AXEL G. J........................ Link Belt Co., Chicago, Ills.
RASHLEIGH, WILLIAM J............................ Aurora, Minn.
RAYMOND, HENRY A.............Rockefeller Bldg., Cleveland, Ohio
REDFERN, JOHN A................................ Hibbing, Minn.
REDNER, A. E.................216 Aurora Location, Ironwood, Mich.
REEDER, J. T.. Houghton, Mich.
REEDER, J. H.. Houghton, Mich.
REIGART, JOHN R................................ Princeton, Mich.
REIFEL, H. T.. Nashwauk, Minn.
REYNOLDS, M. K........................Ridge St., Marquette, Mich.
RICE, CLAUDE T.................1420 Monadnock Bldg., Chicago, Ills.
RICE, JOHN H.. Houghton, Mich.
RICE, C. W...............703 Commonwealth Bldg., St. Paul, Minn.
RICHARDS, WILLIAM J........................ Crystal Falls, Mich.
RICHARDS, MORRIS EARL................... Crystal Falls, Mich.
RICHARDS, WILLIAM J.......................... Painesdale, Mich.
RICHARDS, GUY A.......................... Iron Mountain, Mich.
RIDLEY, FREDERICK WILLIAM.................... Calumet, Mich.
RIGBY, WM. A............. Hartington, R. R. No. 1, Ontario, Canada
ROBERTS, ALTON T............................ Marquette, Mich.
ROBERTS, H. M.........710 Security Bank Bldg., Minneapolis, Minn.
ROHN, OSCAR .... Butte, Mont.
ROSE, ROBERT S................................ Marquette, Mich.
ROSKILLY, JOSEPH .... Virginia, Minn.
ROSSMAN, LAWRENCE A..................... Grand Rapids, Minn.
ROUCHLEAU, LOUIS .....4503 Fremont, Ave., S., Minneapolis, Minn.
ROUGH, JAMES H.............................. Negaunee, Mich.
ROWE, HENRY .... Ironwood, Mich.
ROWE, WILLIAM C.................518 20th Ave., E., Duluth, Minn.
RUMSEY, SPENCER S..............610 Wolvin Bldg., Duluth, Minn.
RUSSELL, JAMES .... Marquette, Mich.
RYAN, JOHN A............................. Iron Mountain, Mich.

SALSICH, L. R.. Coleraine, Minn.
SAVAGE, JNO. A.......................Alworth Bldg., Duluth, Minn.
SCADDEN, FRANK .... Crystal Falls, Mich.
SCHABER, CARL F........................ Box 46, Bessemer, Ala.
SCHACHT, WILLIAM H.......................... Painesdale, Mich.
SCHENCK, CHARLES H........3852 Nicollet Ave., Minneapolis, Minn.
SCHLESINGER, H. J.................................. Milwaukee, Wis.

SCHUBERT, GEORGE P............139 Hubbell Ave., Houghton, Mich.
SCOTT, THADDEUS ....518 Providence Bldg., Duluth, Minn.
SEAMAN, A. E.................................. Houghton, Mich.
SEBENIUS, JOHN UNO................. Wolvin Bldg., Duluth, Minn.
SEEBER, R. R.. Winona, Mich.
SEELYE, R. W........................303 N. 4th St., Brainerd, Minn.
SELDEN, WILLIAM H. JR......................... Iron River, Mich.
SELLS, MAX.. Florence, Wis.
SELLWOOD, R. M.................................... Duluth, Minn.
SENTER, A. W... Hubbell, Mich.
SHELDEN, R. SKIFF.............................. Houghton, Mich.
SHELDON, ALBERT F............112 W. Arch St., Marquette, Mich.
SHERWOOD, M. J................................ Marquette, Mich.
SHIELDS, IRVIN J.................................. Duluth, Minn.
SHOVE, BRIGHAM W............................ Ironwood, Mich.
SIEBENTHAL, W. A.................................. Vulcan, Mich.
SILLIMAN, THOMAS B.......................... Coleraine, Minn.
SILVER, C. R..........................29 W. Lake St., Chicago, Ills.
SLINEY, DAVID J................................ Ishpeming, Mich.
SMALL, H. H.....................4834 S. Halsted St., Chicago, Ills.
SMITH, FRED .... Houghton, Mich.
SMITH, WILLARD J................................ Mohawk, Mich.
SMITH, CARL G................................... Kearsarge, Mich.
SMITH, F. J....................................... Stambaugh, Mich.
SMYTH, H. L..........................Rotch Bldg., Cambridge, Mass.
SOADY, HARRY .... Duluth, Minn.
SPARKS, BENJAMIN F...............205 Ruby St., Houghton, Mich.
SPERR, F. W.. Houghton, Mich.
STAKEL, CHARLES J............................. Republic, Mich.
STANTON, J. R.......................11 William St., New York City
ST. CLAIR, GEO. H..............710 Sellwood Bldg., Duluth, Minn.
STEPHENS, JAMES .... Ishpeming, Mich.
STRONG, CLARENCE G..........Lunkenheimer Co., Cincinnati, Ohio
SUESS, JOSEPH E................................ Negaunee, Mich.
SULLIVAN, A. J................................... Chisholm, Minn.
SUTHERLAND, D. E............................. Ironwood, Mich.
SWIFT, GEORGE D................................. Duluth, Minn.
SWIFT, PAUL D.................................. Houghton, Mich.
SWIFT, H. L.. Houghton, Mich.

TALBOYS, HENRY H............717 Providence Bldg., Duluth, Minn.
TAPPAN, WILLIAM M.............................. Hibbing, Minn.
TARR, S. W........................610 Wolvin Bldg., Duluth, Minn.
THIEMAN, EDWARD .... Florence, Wis.
THOMS, REUBEN KNIGHT .... Ely, Minn.
THOMPSON, CARMI A........Room 222, G. N. Bldg., St. Paul, Minn.
THOMPSON, HENRY S.............................. Beacon, Mich.

THOMPSON, JAMES R.............................. Ishpeming, Mich.
TOWNSEND, C. V. R.............................. Negaunee, Mich.
TREBILCOCK, JOHN .... Ishpeming, Mich.
TREBILCOCK, WILLIAM .... North Freedom, Wis.
TREZONA, CHARLES .... Ely, Minn.
TREVARROW, HENRY .... Negaunee, Mich.
TREVARTHAN, W. J............................. Bessemer, Mich.
TRIPP, CHESTER D...........1835 People's Gas Bldg., Chicago, Ills.
TRUDGEON, JOHN .... Wakefield, Mich.
TRUETTNER, WALTER F......................... Bessemer, Mich.
TUBBY, CHARLES W..613 Merchants Nat. Bank Bldg., St. Paul, Minn.
TUFTS, JOHN W..................900 Hackett Ave., Milwaukee, Wis.
TURNER, CHAS N..............Colby-Abbott Bldg., Milwaukee, Wis.

UHLER, FRED WALTER...............Alworth Bldg., Duluth, Minn.
ULRICH, WILLIAM F............................. Chisholm, Minn.
UREN, WILLIAM J...............38 College Ave., Houghton, Mich.

VALLAT, BENJAMIN W..........347 E. Grand Block, Detroit, Mich.
VAN DYKE, W. D..................910 Wells Bldg., Milwaukee, Wis.
VANDEVENTER, VIVIAN H........................Ishpeming, Mich.
VAN EVERA, JOHN R........................... Marquette, Mich.
VAN EVERA, WILBUR ....Finch Bldg., Virginia, Minn.
VAN MATER, J. A.......................55 Wall St., New York City
VAN ORDEN, F. L.............................. Houghton, Mich.
VICKERS, J. TODD .... Biwabik, Minn.
VIVIAN, JAMES G.................909 Alworth Bldg., Duluth, Minn.
VOGEL, F. A..........................25 Broad St., New York City

WADE, JEPTHA H....................Wade Bldg., Cleveland, Ohio
WAGNER, JOHN M.............................. Houghton, Mich.
WALKER, ROBERT S..................Fidelity Bldg., Duluth, Minn.
WALKER, ELTON WILLARD.............................. Mass, Mich.
WALL, JAMES S.............................. Iron River, Mich.
WALLACE, W. R................................ Houghton, Mich.
WALLACE, GEORGE .... Marquette, Mich.
WARE, JOHN FRANKLIN....Forest and Five Oaks Ave., Dayton, O.
WARE, FRED W................................ Negaunee, Mich.
WARREN, O. B................................... Hibbing, Minn.
WATSON, CHARLES H....................... Crystal Falls, Mich.
WEARNE, WILLIAM .... Hibbing, Minn.
WEBB, FRANCIS J................812 Fidelity Bldg., Duluth, Minn.
WEBB, WALTER M................................. Gilbert, Minn.
WELLS, PEARSON................221 Van Dyke Ave., Detroit, Mich.
WEST, WILLIAM J................................ Virginia, Minn.
WHEELWRIGHT, O. W............................. Florence, Wis.
WHITE, WILLIAM .... Virginia, Minn.

WHITE, EDWIN E....................Sakchi, Bahir and Orissa, India
WHITE, J. W..........................4426 Malden St., Chicago, Ills.
WHITEHEAD, R. G.................................... Alpha, Mich.
WHITTLE, CHARLES E............304 Randolph St., Chicago, Ills.
WILCOX, LEE L.................................... Gilbert, Minn.
WILDES, F. A.................................... Hibbing, Minn.
WILLARD, PAUL D................................ Hibbing, Minn.
WILKINS, WILLIAM .... Ashland, Wis.
WILLIAM, THOMAS H.................................. Ely, Minn.
WILLIAMS, PERCIVAL S........................... Ramsay, Mich.
WILLIAMS, DEAN R...........1213 Majestic Bldg., Milwaukee, Wis.
WILSON, EUGENE B................................ Scranton, Pa.
WILSON, ARTHUR O............................... Hibbing, Minn.
WILSON, W. G..................................... Palmer, Mich.
WINCHELL, H. V..826 First Nat'l.-Soo Line Bldg., Minneapolis, Minn.
WINTER, JOSEPH H.............................. Negaunee, Mich.
WITHERBEE, F. S.......................... Port Henry, New York
WOODBRIDGE, DWIGHT E............Sellwood Bldg., Duluth, Minn.
WOODWORTH, G. L............................. Iron River, Mich.
WOOLF, PERCIVAL J................Monadnock Bldg., Chicago, Ills.
WORDEN, EUCLID P............58 Franklin Place, Montclair, N. J.

YATES, WILLIAM H....................Alworth Bldg., Duluth, Minn.
YOUNG, H. OLIN.................................. Ishpeming, Mich.
YUNGBLUTH, A. J.............................. Ishpeming, Mich.

ZAPFFE, CARL ......213 Citizens State Bank Bldg., Brainerd, Minn.
ZIMMERMAN, WALTER G.......................... Duluth, Minn.

DECEASED MEMBERS.

ARMSTRONG J. F.........1898

BAWDEN, JOHN T........1899
BENNETT, JAMES H.........
BIRKHEAD, LENNOX 1911
BROOKS, T. B............1902
BULLOCK, M. C...........1899
CADDY, THOMAS 1914
COLE, WM. H.............1915
COWLING, NICHOLAS ...1910
CONRO, ALBERT 1901
CLARK, H. S..................
CLEAVES, WILL S........1910
COOPER, JAS. B..........1914
CHADBOURNE, T. L.......1911
CUMMINGS, GEO. P.......1911
DANIELS, JOHN 1898
DEACON, JOHN 1913
DICKENSON, W. E.......1899
DOWNING, W. H.........1906
DRAKE, J. M.............1913
DUNCAN, JOHN 1904
DUNSTON, THOMAS B.......
GARBERSON, W. R.......1908
GAY, JOS. E..............1914
HALL, CHAS. H..........1910
HARPER, GEORGE V.....1905
HASELTON, H. S.........1911
HAYDEN, GEORGE 1902
HINTON, FRANCIS 1896
HOLLAND, JAMES 1900
HOLLEY, S. H............1899
HOUGHTON, JACOB 1903
HYDE, WELCOME ....
JEFFREY, WALTER M...1906
JEWETT, N. R............1914
JOCHIM, JOHN W........1905
KOENIG, GEORGE A......1913
KRUSE, JOHN C..........1907
LINSLEY, W. B...........1914
LUSTFIELD, A............1904
LYON, JOHN B...........1900
MAAS, WM. J.............1911
MARR, GEORGE A........1905
MILLER, A. M............1912
MINER, A. B..............1913
MITCHELL, SAMUEL 1908
M'VICHIE, D..............1906
M'NAMARA, T......... ...1912
NINESE, EDMUND 1909
OLIVER, HENRY W......1904
PEARCE, H. A............1905
PERSONS, GEORGE R....1908
PHILBIN, D. M............1914
POPE, GRAHAM 1912
ROBERTS, E. S...............
ROBERTS, HARRY 1914
ROWE, JAMES 1911
RYAN, EDWARD 1901
SHEPHARD, AMOS 1905
SHERLOCK, THOMAS 1915
SHERRERD, JOHN M......1915
STANLAKE, JAMES 1910
STANTON, JOHN 1906
STEVENS, HORACE J....1912
STURTEVANT, H. B......1910
THOMAS, HENRY 1905
THOMAS, WILLIAM ....
TOBIN, JAMES 1912
TRAVER, WILBUR H......1915
TREVARTHEN, G. C......1898
TRUSCOTT, HENRY 1910
VAN DYKE, JOHN H......1906
WALLACE, JOHN 1898
WHITE, PETER 1908
WHITNEY, J. D...........1894
WILLIAMS, W. H.........1897
WINCHELL, N. H.........1915

LIST OF DECEASED MEMBERS REPORTED SINCE THE ANNUAL MEETING OF 1915.

ABEEL, GEO. H	October 14, 1915
ANDREWS, C. E	July, 1916
CARMICHAEL, WM	November 12, 1915
CARTER, R. B	April 8, 1916
CHYNOWETH, B F	May 10, 1916
CLIFFORD, J. M	January 31, 1917
GARDNER, W. A	May 11, 1916
HOLMAN, J. WINCHESTER	May 10, 1916
MAAS, GEORGE J	December 2, 1915
STANTON, F. M'M	September, 1916

LIST OF PAPERS PUBLISHED IN PRECEDING VOLUMES.

1893—Vol. I.

1902—Vol. VIII.

1903—Vol. IX.

1904—Vol. X.

1905—Vol. XI.

1906—Vol. XII.

1908—Vol. XIII.

1909—Vol. XIV.

1910—Vol. XV.

1911—Vol. XVI.

1912—Vol. XVII.

1913—Vol. XVIII.

1914—Vol. XIX.

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meeting, due notice having been mailed in writing by the Secretary to him.

V.

DUES.

The membership fee shall be five dollars and the annual dues five dollars, and applications for membership must be accompanied by a remittance of ten dollars; five dollars for such membership fee and five dollars for dues for the first year. Honorary members shall not be liable to dues. Any member not in arrears may become a life member by the payment of fifty dollars at one time, and shall not be liable thereafter to annual dues. Any member in arrears may, at the discretion of the Council, be deprived of the receipt of publications or be stricken from the list of members when in arrears six months; Provided, That he may be restored to membership by the Council on the payment of all arrears, or by re-election after an interval of three years.

VI.

OFFICERS.

There shall be a President, five Vice Presidents, five Managers, a Secretary and a Treasurer, and these Officers shall constitute the Council.

VII.

TERM OF OFFICE.

The President, Secretary and Treasurer shall be elected for one year, and the Vice Presidents and Managers for two years, except that at the first election two Vice Presidents and three Managers shall be elected for only one year. No President, Vice President, or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. The term of office shall continue until the adjournment of the meeting at which their successors are elected.

Vacancies in the Council, whether by death, resignation, or the failure for one year to attend the Council meetings, or to perform the duties of the office, shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; Provided, That such appointment shall not render him ineligible at the next election.

VIII.

DUTIES OF OFFICERS.

All the affairs of the Institute shall be managed by the Council except the selection of the place of holding regular meetings.

The duties of all Officers shall be such as usually pertain to their offices, or may be delegated to them by the Council.

The Council may, in its discretion, require bonds to be given by the Treasurer, and may allow the Secretary such compensation for his services as they deem proper.

At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Five members of the Council shall constitute a quorum; but the Council may appoint an executive committee, business may be transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to the approval of a majority of the Council, subsequently given in writing to the Secretary and recorded by him with the minutes.

There shall be a meeting of the Council at every regular meeting of the Institute and at such other times as they determine.

IX.

ELECTION OF OFFICERS.

Any five members not in arrears, may nominate and present to the Secretary over their signatures, at least thirty days before the annual meeting, the names of such candidates as they may select for offices falling under the rules. The Council, or a committee thereof duly authorized for the purpose, may also make similar nominations. The assent of the nominees shall have been secured in all cases.

No less than two weeks prior to the annual meeting, the Secretary shall mail to all members not in arrears a list of all nominations made and the number of officers to be voted for in the form of a letter ballot. Each member may vote either by striking from or adding to the names upon the list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing the ballot with his name, and either mailing it to the Secretary, or presenting it in person at the annual meeting.

In case nominations are not made thirty days prior to the date of the annual meeting for all the offices becoming vacant under the rules, nominations for such offices may be made at the said meeting by five members not in arrears, and an election held by a written or printed ballot.

The ballots in either case shall be received and examined by three tellers appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected. The ballot shall be destroyed, and a list of the elected officers, certified by the tellers, shall be preserved by the Secretary.

X.

MEETINGS.

The annual meeting of the Institute shall be held at such time as may be designated by the Council. The Institute may at a regular meeting select the place for holding the next regular meeting. If no place is selected by the Institute it shall be done by the Council.

Special meetings may be called whenever the Council may see fit; and the Secretary shall call a special meeting at the written request of twenty or more members. No other business shall be transacted at a special meeting than that for which it was called.

Notices of all meetings shall be mailed to all members at least thirty days in advance, with a statement of the business to be transacted, papers to be read, topics for discussion and excursions proposed.

No vote shall be taken at any meeting on any question not pertaining to the business of conducting the Institute.

Every question that shall properly come before any meeting of the Institute, shall be decided, unless otherwise provided for in these rules, by the votes of a majority of the members then present.

Any member may introduce a stranger to any regular meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

XI.

PAPERS AND PUBLICATIONS.

Any member may read a paper at any regular meeting of the Institute, provided the same shall have been submitted to and approved by the Council, or a committee duly authorized by it for that purpose prior to such meeting. All papers shall become the property of the Institute on their acceptance, and with the discussion thereon, shall subsequently be published for distribution. The number, form and distribution of all publications shall be under the control of the Council.

The Institute is not, as a body, responsible for the statements of facts or opinion advanced in papers or discussions at its meetings, and it is understood, that papers and discussions should not include personalities, or matters relating to politics, or purely to trade.

XII.

SPECIAL COMMITTEES.

The Council is authorized to appoint from time to time special committees to consider and report upon, to the Institute through the Council, such subjects as changes in mining laws, safety devices, the securing and editing of papers on mining methods, definition of mining terms, affiliations with other societies, and such other subjects as the Council shall deem it desirable to inquire into, such reports not to be binding on the Institute except action is taken by the Institute in accordance with the rules, and the Council is authorized to expend not exceeding six hundred dollars in any one year to carry out the purpose of this section.

XIII.

AMENDMENTS.

These rules may be amended by a two-thirds vote taken by letter ballot in the same manner as is provided for the election of officers by letter ballot; Provided, That written notice of the proposed amendment shall have been given at a previous meeting.

PAPERS

PAPERS

not only the methods of attacking the orebody, but also the locating and sinking of shafts.

At some period, far back in geological time, it is apparent that the Menominee area was a great granite area; deposited upon the granite was a quartzite, and upon the quartzite was a dolomite, and upon the dolomite a slate. At, or near, the base of the slate was an iron-bearing formation, quite lean, of course, since it was unconcentrated. After these rocks were deposited the earth's crust in the whole Lake Superior district was subjected to enormous pressure which produced intense folding. In the Menominee area the rock formations were thrown up into ridges whose axes extended nearly east and west. Superimposed upon these ridges, or folds, were smaller ones which had others, still smaller, superimposed upon them, and so on, down to the smallest folds. After the folding there was a period of erosion during which some of the ridges were cut to the dolomite, while others were not cut so deeply. This folding and erosion resulted in the exposure of the truncated edges of the inclined formations, the number of formations exposed depending on the depth of erosion. It will not be necessary for the present purpose to follow this geological action further except in so far as it relates to the Chapin mine.

Geological study has shown that the Chapin mine lies on the south side of the most southerly ridge that has been eroded to the dolomite. South of the mine there are no surface exposures other than slate until the greenstones are reached. It has been learned that the Chapin mine lies in one of the smaller westward pitching folds that have been superimposed upon the larger ones. It has been further learned that the folding in the immediate area has been sufficient to push the strata from a horizontal position to a vertical one, and even beyond, so that the dolomite now lies in an inclined position upon, instead of beneath, the slate near whose base is the iron bearing member; that is the geologically older dolomite is on the hanging wall side of the formation, and the dip of the formation is north, despite the fact that it is on the south side of the fold. After being tilted this lean iron-bearing formation was concentrated by underground waters. In some places concentration has been completed but in others it is far from complete even at the present time. Where it is incomplete conditions vary, depending upon the course taken by the concentrating waters, and the degree to which they have completed their work. Where the water has circulated along the footwall there is a layer of ore with a slate foot and a jasper hanging. Where the circulation has been higher up in the lens there is a seam of ore with both foot and hanging of jasper. In other places there are alternating seams of ore and jasper. In still other places are found seams of ore with slate hanging and jasper foot. Occasionally there is a horse of jasper surrounded by ore. Where concentration is complete, or nearly complete, a soft ore which caves readily has been developed. Where concentration is not so nearly complete the ore is leaner and harder, and the particles are bound together more firmly. In the jasper areas, where there has been still less concentration, the material is quite hard and stands quite well.

The members of the formation other than the iron-bearing member have also been acted upon and altered by the folding and the underground waters. While the slate is in places quite firm and unaltered it has in others become quite soft and friable. Where these altered slates form the foot or hanging of the ore body they cause trouble in mining. The dolomite has also been altered. In it are many zones of shearing along which talc has been developed. This is especially so along the contact between the dolomite and the slate which is naturally a line of weakness and, therefore, a plane of shearing. Furthermore, the dolomite has supplied the necessary magnesium to cause the development of a very considerable amount of talc throughout the whole layer of slate between the dolomite and the iron-bearing formation. In view of the fact that talc is prone to swell when exposed to the air and water it is a source of great difficulty where encountered by crosscuts and shafts. The dolomite contains numerous water courses, especially near its contact with the slate. Encountering one of these water courses is a source of very great difficulty, and one that must be kept constantly in mind. Practically all of the water that is pumped from the Chapin mine comes from the dolomite. Any belief that the orebody is wet is incorrect. There is a rather large volume of water to be handled but it is derived largely from the dolomite, and the orebody is, in the main, quite dry. It should be added that the ground above the orebody was originally swamp land in which was considerable water, some of which seeped through be encountered at any time. However, knowledge of the ground, gained from underground operations and from diamond drilling makes it quite possible for the present day operators to choose the best sites for shaft locations.

Shaft sinking has not, on the whole, been a serious problem, the surface being, for the most part, shallow and easily penetrated. However, there was some difficulty in getting "D" Chapin through the surface, which difficulties were overcome by using a freezing process. At the point selected the surface was known to be nearly a hundred feet and was known to be largely quicksand. The method employed was to freeze a wall about the ground through which the shaft was to be sunk and to work under the protection of the frozen wall. Twenty-six 10-in. pipes, located on a circle 29 ft. in diameter were driven to ledge. In each of these pipes was placed a water-tight 8-in. pipe closed at the lower end. The larger pipes were then drawn and the ground allowed to close about the smaller pipes whose distances apart were 3½ ft. from center to center. A freezing mixture was circulated through the pipes causing the ground to freeze. Two qualities were necessary in the mixture used; it had to be a medium that could be made to circulate through the pipes, and it had to have a very low freezing point. The mixture used was a saturated solution of $CaCl_2$ which has a freezing point of -40°. The solution was conducted to the bottom of the 8-in. pipe through a 1½-in. pipe, where it left the smaller pipe and returned to the surface through the larger. This cold solution in the larger pipe abstracted the heat from the adjacent ground and froze it. The solution was cooled at the surface, its heat being absorbed by evaporating ammonia which had been previously compressed to liquid form. After fifteen days of freezing, excavation was begun, and, in 135 days, the shaft had reached ledge, though it was necessary to do some further work under the protection of the frozen wall. The cylinder of frozen ground, in the center of which was the shaft, was found to be more than fifty feet in diameter.

Troubles have at times been encountered in shaft sinking below the surface covering. Talc seams have caused trouble by swelling and crushing the timbers. The story of the sinking of No. 2 Hamilton shaft by Mr. J. T. Jones, and of how it encountered water and filled quickly is now almost ancient history. For a time the shaft remained filled with water, but

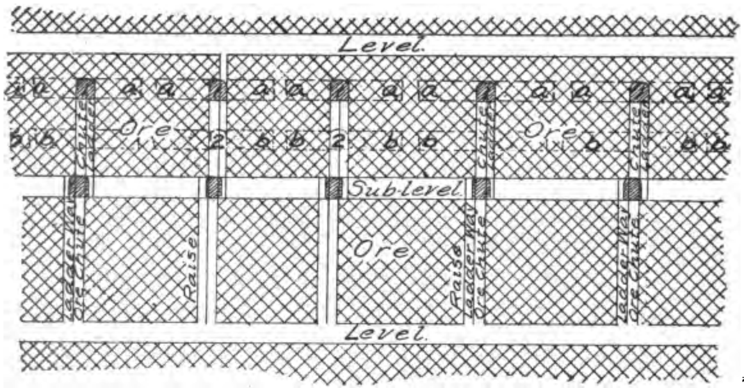


Fig. 1. Longitudinal Section of Orebody Between Two Levels—Showing how preparations for mining are made. Mining will begin in the drifts (a) at the top of each raise, and at the end of each drift farthest from the raise. When the slice of ore above the floor of drifts (1) is mined out the miners begin at (2) in the raise and work as in the slice above. The process is repeated until the level below is reached.

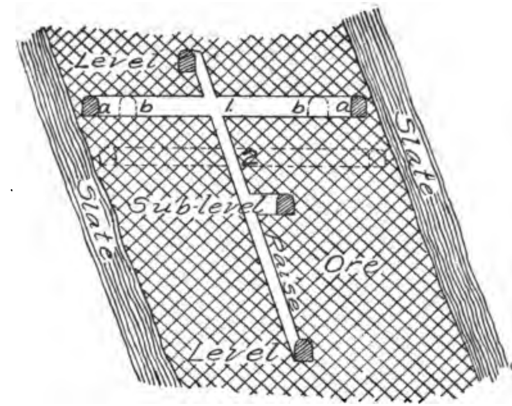


Fig. 2. Cross–Section of Orebody Between Two Levels—Showing how preparations for mining are made. Mining will begin in the ends of drifts (a) farthest from cross–cut (1). When the ore about drift (a) is mined drift (b) will be opened and the ore mined from around it, and like drifts opened until all the ore above the floor of cross-cut (1) is mined. Cross–cut (2) is then opened and mined in like manner. The process is repeated until the level below is reached.

later it was unwatered and used for part of its depth. More recently it has been wholly unwatered and lined with concrete thus making it practically water-tight. It is safe to say that this concrete shaft with its electric pumps is almost a model among present day shafts.

In the first attempts at mining at the Chapin mine the ore was removed from foot to hanging by back-stoping, the hanging being supported by stull timbers. It was soon learned that conditions were such as to require some changes because the hanging slate, being soft and friable, crumbled away from around the stull timbers and allowed them to loosen. To correct this head-boards were placed on the end of the stull timbers against the hanging so as to cover as much of the wall as possible. Even this did not overcome the difficulty. Moreover the vein was becoming so wide that timbers long enough to reach across the excavation could not be obtained. To overcome this new difficulty a layer of ore was left along the hanging and the excavation was made only as wide as timber could be procured to reach across. It was soon evident that the layer of ore left could not be supported, and also that too much ore was being lost by this method of mining. A wholly new method was necessary.

At this stage the "room and pillar" method was introduced. A drift was driven the length of the orebody, along the middle where the ore was wide and near the foot where it was narrower. The ore was then laid out into rooms and pillars, the standard pillar being eighteen feet and room twenty feet, though there were many variations from this standard. In opening a room a drift was driven from foot to hanging the full width of the room. The ground about this excavation was supported by sets made of large square timbers placed eight feet apart from center to center. Each set rested upon a sill which extended two feet into niches cut into the pillar on either side of the room. Both the back and the sides were lagged. A raise connected the back of the room with the level above. After the first cut was taken from the room a second cut was taken from immediately above, and sets of timber placed in a similar manner. In this way cut after cut was taken from the back of the room until the room of the level above was reached, at which point the timbers of the two rooms were connected and the weight taken off the sill of the room above. While a room was being opened the broken ore fell upon the lagging on the lower set of timbers from where it fell to the floor through openings along the side of the room and was shoveled up by the trammers. At a later date chutes were put in and the ore was drawn directly from above the lagging into the tram cars. By this method room below room were joined together with the result that rooms and pillars were constantly growing higher.

It can be readily seen that two serious difficulties were soon to be encountered in this method of mining. In the first place a vast amount of ore was being left in pillars which must be recovered. In the next place it was evident that the pillars would soon crush under their own weight; in fact, crushing had already begun. Still ignoring the first diffi-

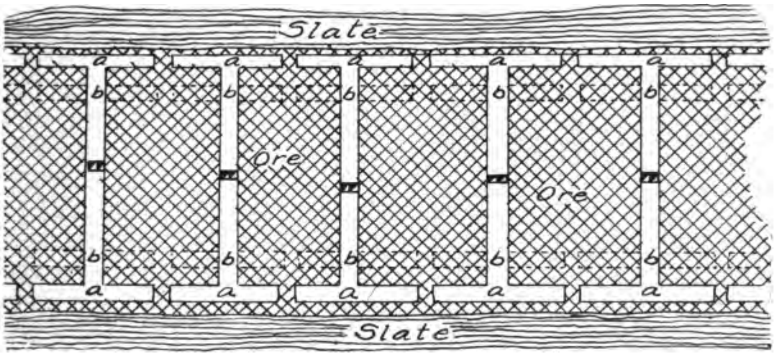


Fig. 3. Horizontal Section Through the Orebody at the Base of a Slice of Ore to be Removed—Showing how the ore is to be mined. When the ore about drift (a) is mined drift (b) will be driven and the ore about it mined, and the process repeated until all the ore back to the raise is removed, after which the whole operation is repeated about sixteen feet below.

culty an attempt was made to correct the second by using square sets, the timbers of which were cut in lengths of six feet and framed together. But even these were not sufficient to hold the ground. There was soon such an increased amount of crushing that the surface began to give way. It became imperative that some method be introduced that would support the pillars until such time as the ore could be recovered from them. Furthermore, it was not advisable to open more rooms. It had come to a time when new methods were an absolute necessity.

The "room and pillar" method was abandoned and new methods introduced which, it was hoped, would make it possible to recover the ore left in the pillars, and also to mine the other ore without encountering the difficulties that had heretofore been encountered. Accordingly, filling of the old rooms was begun. The first filling material was the surface covering of the ore body which was stripped off and dumped into the rooms through a small shaft in the back of each. Sandstone was used to complete the filling. In this manner each room was filled, and a room thus filled became in itself a pillar which, it was hoped, would support the ground while the original pillar of ore was being removed.

The pillar was approached by a crosscut driven from a drift in the footwall slate parallel to the ore body. The drift was continued through the pillar close to the lagging of the sandstone-filled room. A cut about as wide as a drift and running with the formation was then taken from the base of the pillar along the hanging side. The opening from which the ore was thus removed was filled to the back with sandstone which was brought down from the surface and shoveled into it. Another cut was then taken from alongside of, and on the same level with the first cut, and the opening similarly filled with sandstone. When a cut had in this manner been made across the base of the pillar of ore, a second slice was taken in a similar manner, the broken ore falling upon planking which was laid upon the sandstone filling, and from which it was shoveled. In this manner it was hoped to remove the pillar. The timber and sandstone required in the work were received through a raise in the pillar. Only a small amount of ore was removed in this manner because the ground was heavy, and the amount of filling required was great.

To recover the ore from the parts of the orebody that had not been attacked by the "room and pillar" method a drift was driven in the footwall slate parallel to, and only a few feet distant from the orebody. At the same time a similar drift was driven in the orebody. The two drifts were connected from time to time by crosscuts. The drift in the slate was the main haulage way, while the other was used as an opening from which the ore was attacked. The ore was, in all cases, removed by back-stoping, and the stopes were filled with sandstone quarried at the surface and sent underground. Such rock as was taken from the drifts and other dead workings was also used for filling. It can be readily seen that it

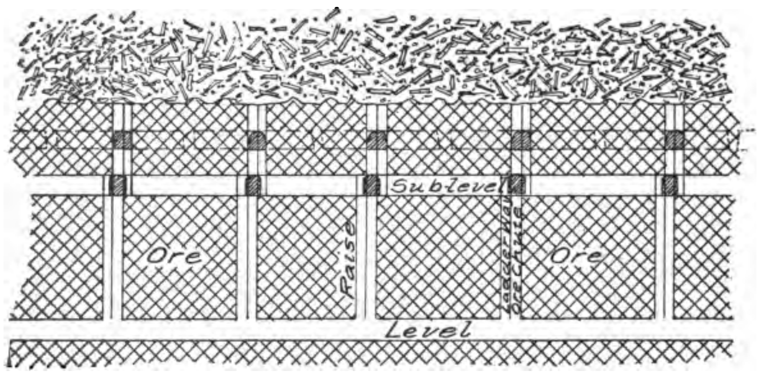


Fig. 4. Longitudinal Section of Orebody—This is the same section as shown in Fig. 1, but the ore about the main level above, and, also, that about sub-level (1) has been mined. Above the ore is a thick covering of lagging and other waste.

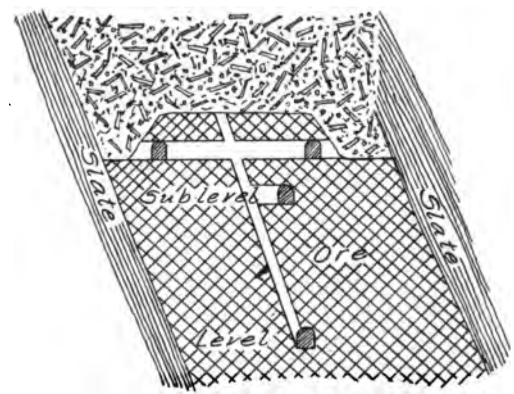


Fig. 5. Cross-Section of Orebody—This is the same section as is shown in Fig. (2), but the ore about the upper level, and that about sub-level (1) has been mined, as has also a part of that about sub-level (2).

took a very considerable force of men to quarry the sandstone, load it into cars, send it underground and distribute it in the stopes. Where the orebody was not wide and was firm it was attacked by longitudinal stoping, but elsewhere it was attacked by transverse stoping, that is the cuts were made across the strike of the formation. In each case the broken ore fell upon planking laid upon the sandstone filling which was always kept near the back. In the transverse stoping gangs of men were started at work fifty feet apart in the ore drift, and each gang drove a crosscut from foot to hanging, eight feet high and nine feet wide. The opening made was filled wtih sandstone, and a similar cut taken from alongside of it, and that opening was also filled. In this manner the work on the level was continued until completed. As the sandstone filling was put in planking was laid upon it for the broken ore to fall upon. A second slice was taken from the back of the stope in the same manner as the first was taken, and the opening filled with sandstone. This work of slicing and filling was continued until the level above was reached. If, at any time, a part of the back appeared weak it was supported by a prop. The sandstone used for filling was sent from the level above through a winze. The ore reached the main haulage way through a raise put up from the side of the slate drift and connected with the stope at the working elevation by a crosscut. The ore was trammed from the stope through the crosscut and dumped into the raise, the enlarged bottom of which served as a pocket from which it was drawn into cars. These stopes were large openings reaching from foot to hanging and for some distance along the strike. Beneath this immense back the miners were at work breaking ore and the fillers were tramming sandstone from the chutes.

It can be readily seen that this "filling" method had a great advantage over the "room and pillar" method in that there was a much smaller timber cost, and, what was more important, no ore was left behind as pillars. On the other hand it necessitated an enormous amount of dead work because of the drifting and crosscutting in the slate and the still greater amount of work necessary to furnish the sandstone filling. Moreover, the ground not only in the old pillars but also in the main stopes was very heavy. It was evident that a better and less expensive method of mining was needed.

By this time the nature of the Chapin orebodies had been pretty well learned, and some facts had been pretty well established. It was very evident that the ore was soft and prone to cave. It was also thoroughly evident that it was beyond the power of man to support it. It was further evident that not only would the under-cut ore cave, but also the adjacent slate would cave, and ultimately all the surface underlain by the caving ore and slate would cave. Any shaft in, or surface structure upon, this caving ground would, sooner or later, become valueless. Any method of mining that did not take all these facts into account could not be cheap and efficient. If a part of the surface was to cave it was advisable to remove all surface structures from that part and abandon it. Likewise, if the ore could not be supported when under-cut it was advisable not to under-cut it and thus make it necessary to attempt to support it. Indeed, it would be desirable, if possible, to adopt a method in which the caving tendency of the ore would assist rather than retard the work. The present methods were adopted in the hope that they would answer all these requirements, and they have served the purpose well.

The working shafts are located in either the hanging or the footwall, and the orebody is approached from these shafts by drift or crosscut as the case may be. The orebody is opened by a drift in the ore running throughout its whole length. This drift is the working level to which all the ore between it and the next level above is sent, and along which it is trammed to the shaft and hoisted to the surface. Experience has taught that it is not advisable to make the level openings more numerous or more elaborate than actually required because the constant crushing of the timbers makes a large amount of repair work necessary. Levels are laid out a hundred feet apart. The work of developing each succeeding lower level is planned, in so far as possible, so that the new level will be ready for mining work when the ore is exhausted from the higher level. After the level is laid out raises are put up from the side of the drift to within about eight feet of the level above, but such of them as are required are holed through to aid ventilation or acquire traveling ways. The raises are generally in ore throughout their whole length, but there are cases where it is necessary, or expedient, to have a part of a raise in slate. At the time the raises are being put up a sub-level is driven midway between the two levels, and connected with the raises by crosscuts. This sub-level affords easier communication between the stopes, and facilitates the work of getting timber and supplies to where they are used.

Near the tops of the raises is where the openings are made for the actual mining of ore. The number of men that can be put at work depends upon the number of raises in which work is being done. Every additional raise entails an additional expense for opening and maintaining. Evidently then it is desirable to have only as many raises as are necessary to work the number of men desired. The best practice seems to be to place the raises from thirty-five to fifty feet apart. At a point about fifteen feet below the floor of the level above a crosscut is driven from each raise to both the foot and the hanging of the ore. At each end of each crosscut a drift is driven in each direction in the ore. The drift along the foot is so located that the bottoms of the outside legs of the sets rest upon or close to the slate, while the hanging drifts are so located that the outside ends of the caps touch, or nearly touch, the hanging-wall. If we assume that the raises, and therefore the crosscuts, are fifty feet apart, then the drifts along either the foot or the hanging will meet when they have been driven twenty-five feet from the crosscut. However they are not driven to meet, a thin wall of ore being left. It is at the end of this short drift that the actual stoping of ore begins. As the miner stands at the end of the drift there is a thickness of eight or ten feet or ore in the back above him. He begins work by attacking the ore at the end of the drift. He fills his car as often as possible, and takes the ore back and dumps it into the raise. When he can get no more ore he shoots out the last set that he put up in the drift, which allows the ground about it to fall from the sides and back, generally of its own weight, but, if necessary it is shaken loose with dynamite. It is then shoveled up and trammed to the raise. Set after set is shot out until all of the ore back to the crosscut is mined. In like manner the ore on the other side of the drift is mined. Similar work is done at the end of the crosscut that has gone from the raise to the opposite wall of the orebody. As the ore is thus removed the thick covering of broken lagging and other waste material that has collected above it settles down and fills the space from which it has been removed. This waste material covers the ore like a mat and keeps it clean. When the miner has taken the ore from around the first drift and has drawn back to the crosscut he makes a second drift just as he made the first except that it is a few feet nearer the raise. The ore is removed from around the second drift in the same manner as it was removed from around the first. The operation is repeated until all the ore is mined back to the raise. At such time as best facilitates the work a second crosscut is driven from the raise about sixteen feet below the one being mined out. The work of crosscutting, drifting and mining is repeated in the same manner as it was done immediately above. In like manner work is done about all the raises. As mining continues the orebody is lowered and the lower level is approached in cuts of about sixteen feet each. When the main level is reached the ore is mined from about it in exactly the same manner as it was mined from other parts of the orebody; the level really becomes a sub-level when the ore about it is being mined. The nature of the ore makes it advisable to remove the ore between the two levels in about six slices, that is about sixteen to seventeen feet to the slice, which means that from eight to ten feet of ore is drawn from the back of the drift. An attempt was made to divide the hundred feet into four parts but the results were not satisfactory.

The orebody was opened on some of the main levels by two main drifts in the ore, one along the foot and one along the hanging, these being connected by crosscuts. It was hoped that this method would enable a more advantageous distribution of the working raises, and would facilitate haulage on the level. The objection was that the numerous openings in the orebody made too much repair work necessary.

The methods described above are the methods theoretically followed, but variations are always made to meet such exceptional conditions as might be encountered. In the work of removing the pillars from the old workings it is often difficult to select locations for raises.

It is not necessary to support the ground around many of the openings for any great length of time; consequently most of the timber used is small. However, those drifts in the ore which have to be held for some time require considerable repair work, even with the heavy timbers that are used; but they are not made larger nor more elaborate than is necessary. Ultimately they too are caved down and mining operations removed to a lower level. No pillars of ore are left behind; no ground is left in such a way that it is liable to cave and ruin the workings; and no great amount of heavy timber is used as with the "room and pillar" method; no great amount of dead work is done as with the "filling" method, nor is a large body of heavy ground under-cut. There are some timber costs, and some dead work is done, but both are small in comparison with past methods. Moreover, little or no ore is lost.

THE BLOCK-CAVING SYSTEM USED AT THE PEWABIC MINE.

BY A. J. MYERS, IRON MOUNTAIN, MICH.*

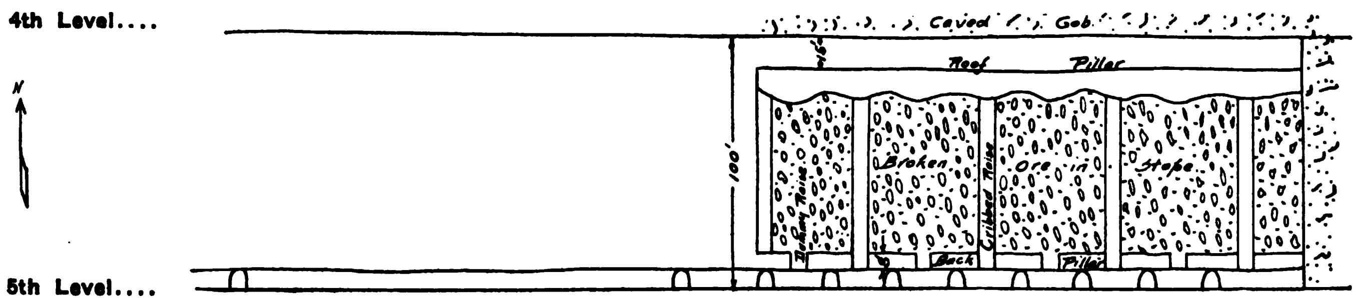
The formation at the Pewabic mine in which this block-caving system is used is about 250 feet wide and stands up straight at 90 degrees from the horizontal. The footwall is the gray Traders slate and the hanging is the Red Brier slate; both walls are uniform in dip and strike. A horizontal sandstone capping, about 100 feet thick overlies the ore formation and forms a strong arch over it. A depth of 500 feet was reached before the sandstone formation broke through to the surface with a hole about 5 feet in diameter, which has since enlarged about 700 feet wide by 1,500 feet long.

There are four distinct operations in this system of mining: namely, developing, caving, opening up and actual mining.

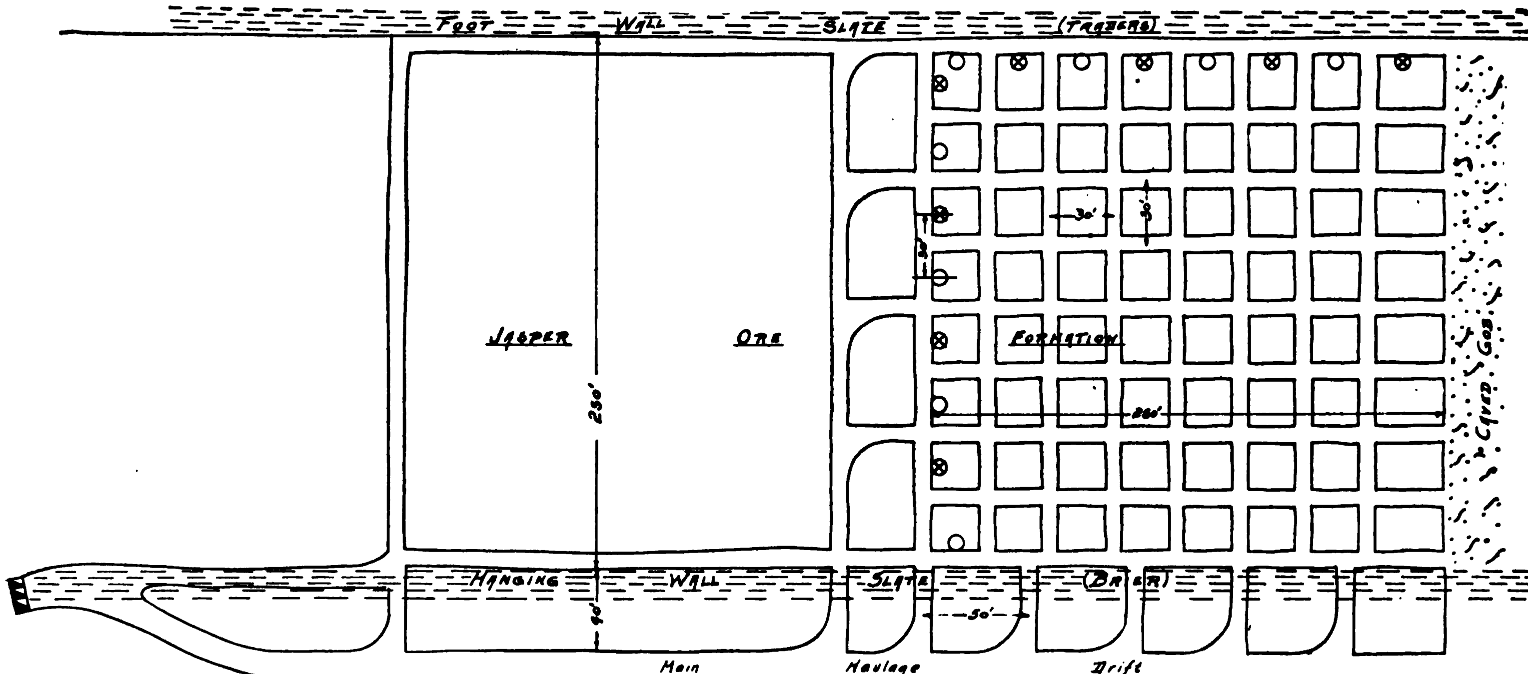
Developing.

The developing of a block consists of driving footwall and hangingwall drifts about 7 feet wide by 8 feet high along the whole length of the block, which is about 250 feet square. Crosscuts and drifts on 30-foot centers are driven north from the hangingwall drift and east from the main crosscuts, the width and length of the block being developed. (See plan sketch). Stopes are then started above the footwall drift and the crosscut at the west end of the block, a back pillar about 6 feet thick being left above the drift and crosscut to hold the broken ore on which the men work while raising the stopes. Raises on 30-foot centers are put up through this back pillar, every other one being cribbed to allow for passage of men, tools, etc. The other raises are known as "dummies" and are used for drawing out the surplus ore broken in the stopes. These stopes are raised to within 20 feet of the level above, a

*Mining Engineer, Pewabic Mining Co., Iron Mountain, Mich.



SECTION THROUGH FOOTWALL STOPE



PLAN OF BLOCK

floor pillar being left to hold back the waste, gob and broken sandstone from the surface. When the stopes have reached the required height, the pillar above the crosscuts and footwall drift is drilled and blasted out, the north and east corner of the block being blasted first at this mine. The stopes are then cleaned out from foot to hanging and from end to end of the block. Thus at the close of the development stage, a block about 250 feet square and from 100 to 125 feet high is cut off on all but the hanging wall side and stands on solid pillars about 22 feet square. In the meantime a main haulage drift has been driven in the red-slate wall parallel to and about 40 feet distant from the ore.

Caving.

The pillars are now weakened by drilling and blasting off the corners; some of the pillars will crush when 6 feet in diameter and others have to be blasted out entirely, depending on the physical character of the ore in the pillar itself. This weakening of the pillars is started on the side of the block farthest from the main haulage drift and proceeds until the width of the formation is covered. In most cases in this mine the north side of the block comes down before the weakening of the pillars is completed to the south side. When the whole block is down, it has dropped about 8 feet and has broken up fine enough to be shoveled into cars.

Mining.

Crosscuts have been driven, meanwhile, from the main haulage drift on 50-foot centers to the south side of the block. From these points timbered drifts are now poled through the broken ore to the north or footwall side, where drifts are driven off at right angles east and west. The drawing out of the ore is now begun, and it is continued at any one point until the sandstone or surface material moving with the ore makes the product too low in iron content. Plank sollars are used and one miner can take care of one or two pairs of trammers, depending on how well the ore has broken in caving. This drawing is continued toward the main haulage drift until the ore is extracted. At first glance one would think that considerable timber is used in poling through the ore; however, only enough crosscuts are driven to supply the output required, and thus only a small portion of the caved block is on timber at any one time.

Several attempts have been made to arrive at the probable loss in mining with this system. The results show that the loss is small and that the system in this respect compares favorably with other systems used in underground iron mining.

THE METHOD OF MINING AT THE LORETTO MINE.

BY C. H. BAXTER, LORETTO, MICH.*

The principal orebody at the Loretto mine is a vein-like deposit dipping almost vertically and averaging about 1,000 feet in length. It is usually from 10 to 40 feet wide, but in places it narrows to a few feet or widens out to 100 feet. For the greater part of the length of the orebody the ore came up to the overburden, which is about 25 feet in depth. The orebody is developed to a depth of 800 feet.

The ore varies greatly both in iron content and in physical characteristics. In some sections it is hard and lean; in others, though of high iron content, it will stand up well; in others the ore is composed of uncemented grains and though easily handled when dry, is almost impossible to hold when wet. The orebody is for the most part dry, but in places water comes in from the wall rocks and makes trouble in mining.

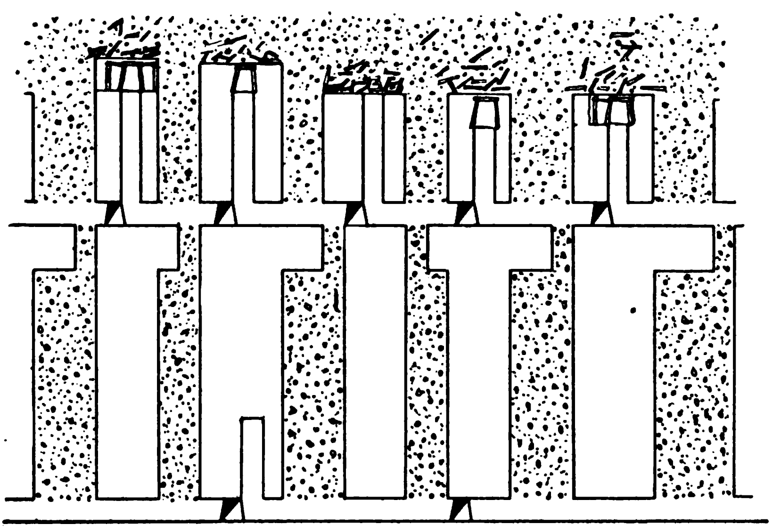
Both foot and hanging walls are very weak slate-like formations with a very marked cleavage with the bedding, and joints across the bedding. The wall rock breaks off first into slabs and then into small angular pieces along the joint planes.

When this orebody was first opened, the Sturgeon River crossed it at about the middle of its length, making necessary the supporting of the surface. The first method of mining was the room-and-pillar method. The distance between levels was 100 feet. On each level a crosscut was driven from the shaft in the footwall to the orebody, and a drift driven longitudinally in the orebody and timbered with square-set timbers. Except under the river, where a large pillar of ore was left to insure the support of the surface, alternate stopes and pillars were laid out; the stopes were usually 2 or 3 sets wide and the pillars 3 or 4 sets wide and extended from foot to hanging. The stopes were mined by the usual square-

*Superintendent, Loretto Mine, Loretto, Mich.

set method to within 20 feet of the level above. By this method, the larger portion of the ore was left in the mine to support the surface.

In 1908 the river was diverted around the orebody* and the removal of the pillars by the top-slicing method commenced. All the stopes in the mine that were open were filled with rock. On each level as mining proceeds a drift is driven in the footwall parallel to the orebody and a crosscut into each pillar. A two-compartment raise, 4 ft. by 7 ft. in outside di-



Typical Longitudinal Section—The old "room and pillar" stopes were mined up, two or three sets long and as wide as the orebody, and a floor pillar left to protect the level above. The sketch shows typical rooms and pillars, and the ore above the upper level being mined in 11 foot slices. At the same time, the lower level is developed, and raises put up through the pillars.

mensions and built of 6-in. cribbing, is put up in each pillar, preferably on the footwall. Mining then proceeds from the top by the usual top-slicing method. Nine slices are taken to a level, each slice thus being about 11 ft. high. In slicing, the ordinary type of drift set is used, with a post 9 ft. long and a cap 4 to 6 ft. inside the joggle, to suit conditions. The timbers are from 6 to 10 in. in diameter. The usual slice is 9 feet wide.

The usual procedure from the top of the raise is to crosscut

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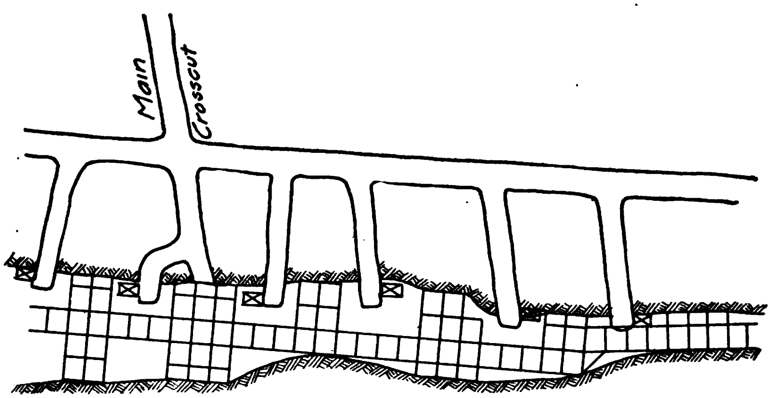
to the hanging wall, drift east and west on the hanging to the filled stopes, then drop back toward the foot by successive side slices until the floor is all removed. The ore is shoveled into ¾-ton buggies and dumped in the raises. As the ore is taken out, the bottom is covered with 1-in. hardwood boards or lagging, and whenever the timber from that portion of the slice from which the ore has been removed shows much weight, it is blasted and the back comes down. Before the sets around the raise are blasted, the slice below is opened up enough to make room for the buggy and tools. As the work advances, the ore on each slice is taken out right up to the floor of the slice above; thus no ore is left to be pulled coming back.

Many variations of the method given above are practiced,

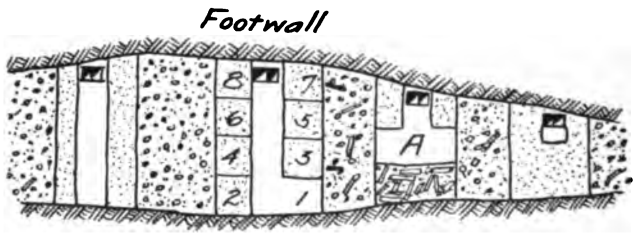


General Level Plan—This shows the old square set workings, the stopes opened from foot to hanging and the drift carrying through. The new main drift is driven about 40 ft. back in the foot and crosscuts put in to the pillars. Two-compartment raises are put up on the foot to the level above.

always in accordance with the general rule that the ground around the raise is weakened as little as possible and that the order of mining is retreating toward the raise. As the pillars vary greatly in size, often more than one raise must be put in a pillar to keep it from lagging behind the others; in this manner the mining is kept at about the same elevation in all the pillars. Since many of the pillars are small, the amount of ore mined from each raise is much less than would be in the case with a new orebody.

On the upper levels, the mat of timber above the back followed the mining down pretty well and in most cases only a few lagging were needed to cover the bottom of the slice. At present, however, the pressure from the walls seems to hold the mat up and the opening left by mining fills with wall rock. If any opening is left in the back, fine rock will run until stopped; this necessitates boarding down each floor very closely.

The amount of dead work in driving drifts and crosscuts in the footwall is very great for the amount of ore mined; but this cannot be avoided. as the settling of the pillars makes it impossible to hold an opening in the orebody for more than a short time. This settling even affects the footwall, so that



Plan of Typical Pillars—Showing the raises near the foot, the crosscut to the hanging and the order of slicing back. The pillar marked "A" shows the hanging slice blasted down.

in places the crosscuts to the orebody must be regraded and retimbered several times a year. The settling of the pillars and old filled stopes in mass has amounted to from 20 to 30 feet for a considerable length of the orebody.

The pillars have now been extracted down to about the fifth level; since each foot of ground is opened up and timbered, the extraction is practically 100 per cent.

MINING METHODS IN THE IRON RIVER DISTRICT OF MICHIGAN.

BY RUDOLPH ERICSON, IRON RIVER, MICH.*

BALTIC MINE—*Operated by the Verona Mining Co.*

A sub-level stoping method is used, with sub-levels laid out 25 feet apart and the ore on each sub-level blocked out so as to leave small pillars. Raises or mills are put up from the lower sub-levels, and the last ore blasted from the back and small pillars goes directly into the mills. As the mills are only 20 feet apart and flared at the tops, very little loose ore is left behind to be mined with the next sub below. With this method of mining, so little tramming is required that it is usually done with wheelbarrows. The mine has a strong jasper hanging.

BATES MINE—*Operated by the Bates Iron Co.*

This mine is worked by a shrinkage-stope method, very similar to that in use at the Hartford mine at Negaunee, and described on page 133 of Volume XIX of the Proceedings of the Lake Superior Mining Institute.

BENGAL MINE—*Operated by the Verona Mining Co.*

The mine is comparatively new and is operated by the top-slicing system. One reason for this, probably the leading one, was the heavy overburden of sand in direct contact with the orebody.

BERKSHIRE MINE—*Operated by the Brule Mining Co.*

The top-slicing system is used in part of this mine, and a modified sub-level stoping method in the rest of it.

CASPIAN MINE—*Operated by the Verona Mining Co.*

The top-slicing system is used throughout. The method is fully described in a paper by Mr. W. A. McEachern, published in Volume XVI, page 239 of the Proceedings of the Lake Superior Mining Institute.

*Superintendent Davidson Ore Mining Co., Iron River, Mich.

CHICAGON MINE—*Operated by the Munro Iron Mining Co.*

In choosing the method of mining at this mine, the operators were confronted with the problem of mining a low grade ore cheaply. The ore is close to or lies directly upon a black slate which has a dip of 70 degrees. The hanging wall is a hard jasper which stands well, being harder than the ore itself.

The method of mining is similar to that employed at the Hiawatha mine, which is described later, except that the levels are 200 feet apart, with five subs equally spaced between them. The ore from the higher subs breaks into small chunks as a result of dropping from a considerable height; this makes tramming and drawing the chutes less expensive than it otherwise would be. A certain amount of ore must be left on the black slate footwall; otherwise the ore will be contaminated by slides of the slaty material.

DAVIDSON NO. 1 MINE—*Operated by the Davidson Ore Mining Co.*

A sub-level system is used, with the sub-levels laid out 20 feet apart. The principal orebody dips to the south and has a flat pitch to the west. The sub-levels are driven east to the end of the ore; raises are then put up from the lower sub-levels and the tops are widened. The ore in the back and sides of the upper sub-level, on the extreme end first, is then blasted directly into the mills and trammed on the lower sub-levels to the main raises leading to the haulage level. The accompanying drawing shows a crosscut with the layout of the pocket at the shaft.

DOBER MINE—*Operated by the Oliver Iron Mining Co.*

The deposit is fairly regular, dipping from 54 to 58 degrees, and is approximately 100 feet wide. The footwall is of black slate, which on one end of the deposit swings around for a short distance and forms a hanging. It was at this point that the broken slate gradually accumulated and caused the Dober mine fire; the high sulphur content of the black slate will produce spontaneous combustion when the slate is broken up and subjected to weight. The hanging wall for the rest of the mine is a flinty jasper of extraordinary hardness. The mining method adopted is shrinkage-stoping. When the stope is within 10 or 15 feet of the level above, the work is transferred to this level, the ore being broken down into the stope below, where it is drawn off as desired. The work on the level above is begun by drilling holes on the back, sides and bottom on the inside end of the ore.

FOGARTY MINE—*Operated by the Verona Mining Co.*

Mining is carried on by means of the sub-level caving system, with levels 100 feet apart and subs 25 feet apart.

FORBES MINE—*Operated by the Jones & Laughlin Ore Co.*

Three methods of mining are used at this mine; namely: top-slicing, sub-level underhand stoping, and shrinkage stoping.

The top-slicing system is very similar to the method in use at the Davidson No. 2 mine, to be described later.

The sub-level underhand stoping is conducted as follows: First, the block of ore which is to be removed is selected. The blocks are usually 70 to 80 feet in length on the vein and include all the ore between the foot and hanging wall. From the main drift on the hanging or foot, as the case may be, and 20 feet to either side of the block of ore, a raise is put up to the top of the ore which is to be mined, or from level to level as the case may be. Two parallel crosscuts with 40-foot centers—the first one 40 feet from the center of the raise —are run from the main drift to the foot or hanging directly under the block of ore. The crosscuts are two sets high; the first so as to permit tramming, and the second so as to allow the fillers to work. The first set is covered with short rails and poles to be used as stoppers in filling. From both sides of the upper set and at close intervals, incline raises are put up on an angle of about 50 degrees. These raises are bell-mouthed at the top; thus the ground above and between the two crosscuts is left as a wedge-shaped pillar. At the top of the small incline raises, vertical raises about 15 feet high are put up, these being laid out in squares 20 feet apart. These save tramming while the sub-level is being driven. The sub-level is driven from the main raise into the ground which is to be stoped at an elevation which will bring the back of the sub-level in the same plane as the tops of the small vertical raises. When the sub-level is completely opened, raises are put up directly over the previously mentioned vertical raises to the point where the next sub-level will be opened. The next operation is blasting down of the bottom of the first sub-level, at the point farthest from raise first, retreating toward the main raise. The same procedure is followed on the next sub-level above.

The shrinkage-stope method is used at only one stope in the mine. Here the orebody is about 50 feet wide. A main haulage drift was driven on the footwall and a parallel drift opened along the hanging. The first raise was put up from the main haulage drift to the top level. Small untimbered raises, 20 feet apart center to center, were next put up from the foot and hanging drifts. At a point 15 feet above the level, the small raises were connected and the stope opened to full size of the ore deposit. From this point on, back-stoping is practiced, only sufficient ore being drawn off to allow working room until the stope is finished, when the ore is drawn off as desired.

HIAWATHA MINE—*Operated by the Munro Iron Mining Co.*

The first requirement in the method of mining to be used at the Hiawatha mine is a cost which is low in keeping with the grade of the ore. In the upper part of the mine the ore is surrounded by a rock and ore formation, but at depth a soft black slate appears on both foot and hanging, making mining under any system very difficult. The orebody is faulty. The ore is hard and tough, breaking in large chunks. The orebody is too narrow for slicing and the grade of the ore prohibits the use of timbers. Back-stoping was tried, but with indifferent success; the broken ore seemed to cement together and did not draw down, and it was very apt to bridge over and then fall when the men were on top.

Sub-level stoping was finally adopted, the method being as follows: The new level is driven 135 feet below the old level and is timbered 7 feet in the clear. Between the two levels three subs are driven at equal distances and the lower sub is connected with the main level by raises 24 feet apart. The lower sub, at the far end of the orebody is stoped back perhaps 20 feet for the width of the orebody, a bottom of ore being left against the sub above. The second sub is then worked back until its back and bottom are flush with the face of the stope below; and so on with the other subs.

To break the ore in the first sub costs 25 to 50% more than in the upper subs. The method requires no timbering. The men are always working under cover, never in open stopes, The only essential precaution is to leave a layer of ore on the black rock to keep it from sliding into the stope before the ore is all removed. One disadvantage of this method of mining is that owing to the extreme toughness of the ground the ore breaks into large chunks which have to be blasted to pass the chutes. The expense of this is considerable; a softer ore would break up readily and would be much cheaper to mine.

HOMER MINE—*Operated by the Buffalo Iron Mining Co.*

The main orebody in the Homer mine is a rather hard limonitic hematite. It varies in width from 20 to 100 feet and dips from 45 to 60 degrees. These conditions are best adapted to sub-level stoping.

The angle of the dip would make back-stoping difficult, and the hardness of the ore makes possible practically as high a percentage of extraction as with a caving system and at a lower cost per ton.

The preliminary raises are started from the haulage level and put up about 200 feet apart in the ore. They are about 4 feet square, and are not timbered. As soon as these raises are high enough so that there is 14 feet of solid material above the haulage level, a sub-level is started from both sides of the raise. The raise is then continued and sub-levels started from it in the same manner about every 20 feet vertically. The sub-drifts started from these raises are driven until they meet similar sub-drifts started from other such raises. Where the orebody is less than 30 feet wide, only one such drift is needed; but where it is wider two drifts are put in and connected with crosscuts every 100 feet. Auxiliary raises are put up from the haulage level to these sub-levels every 50 feet, to reduce the distance the ore must be wheeled in the sub-levels. The first step in mining the ore between the subs is the bell-mouthing of the top of one of the raises from the main level to first sub-level. The back of the first level is then broken directly into the raise. This is continued clear through to the top sub. The stopes are widened by breaking the ore on each sub in succession into the open stope, which is thus gradually increased in size. The men are at all times a short distance away from the open stope and have a back of solid ore directly overhead. The stopes are opened up until they are as large as is thought practicable and pillars are left to keep the hanging wall from caving. With the orebody from 40 to 60 feet wide, these stopes are made about 150 feet long.

When the stopes have been completed, raises are put through the pillars and as large a part of the ore as possible is blasted into the stopes on each side and drawn out. These pillars are from 25 to 50 feet in width, depending on where they are left and what they have to support. Where the footwall is as flat as 45 degrees, the ore does not slide readily; hydraulicking with a 2-in. stream has been tried with considerable success.

JAMES (OSANA) MINE—*Operated by the Mineral Mining Co.*

A system of sub-level stoping is employed. The orebody is overlaid by a jasper capping and does not extend up to the sand.

ROGERS MINE—*Operated by the Munro Iron Mining Co.*

The final method of ore extraction has not yet been chosen. The unusually large amount of water which the mine makes has required extensive drainage openings and has precluded for the time being any extension of workings into direct contact with the overburden.

SPIES MINE—*Operated by The Cleveland-Cliffs Iron Co.*

The orebody is now being developed; the method of mining will be decided upon later.

TULLY MINE—*Operated by the Corrigan-McKinney Co.*

Two methods of mining are employed, sub-level stoping somewhat similar to that in use at the Hiawatha mine, and shrinkage-stoping.

YOUNGS MINE—*Operated by The Huron Mining Co.*

A method combining shrinkage-stoping and sub-level stoping is used.

ZIMMERMANN MINE—*Operated by the Spring Valley Mining Co.*

An attempt was made to adopt the block-caving system of mining but it was found impossible to bring down the ore uniformly without contamination from the overburden and the method had to be abandoned. At the present time several of the regular methods are being tried.

For the above data regarding local methods, the writer is especially indebted to the following:

Mr. D. H. Campbell, Superintendent, Munro Iron Mining Co.

Mr. H. S. Peterson, Chief Engineer, Jones & Laughlin Ore Co.

Mr. W. A. McEachern, Mining Engineer, Verona Mining Co.

Mr. R. C. Mahon, Mining Engineer, Wickwire Mining Co.

THE METHOD OF OPENING AND MINING THE DAVIDSON NO. 2 MINE.

BY RUDOLPH ERICSON, IRON RIVER, MICH.*

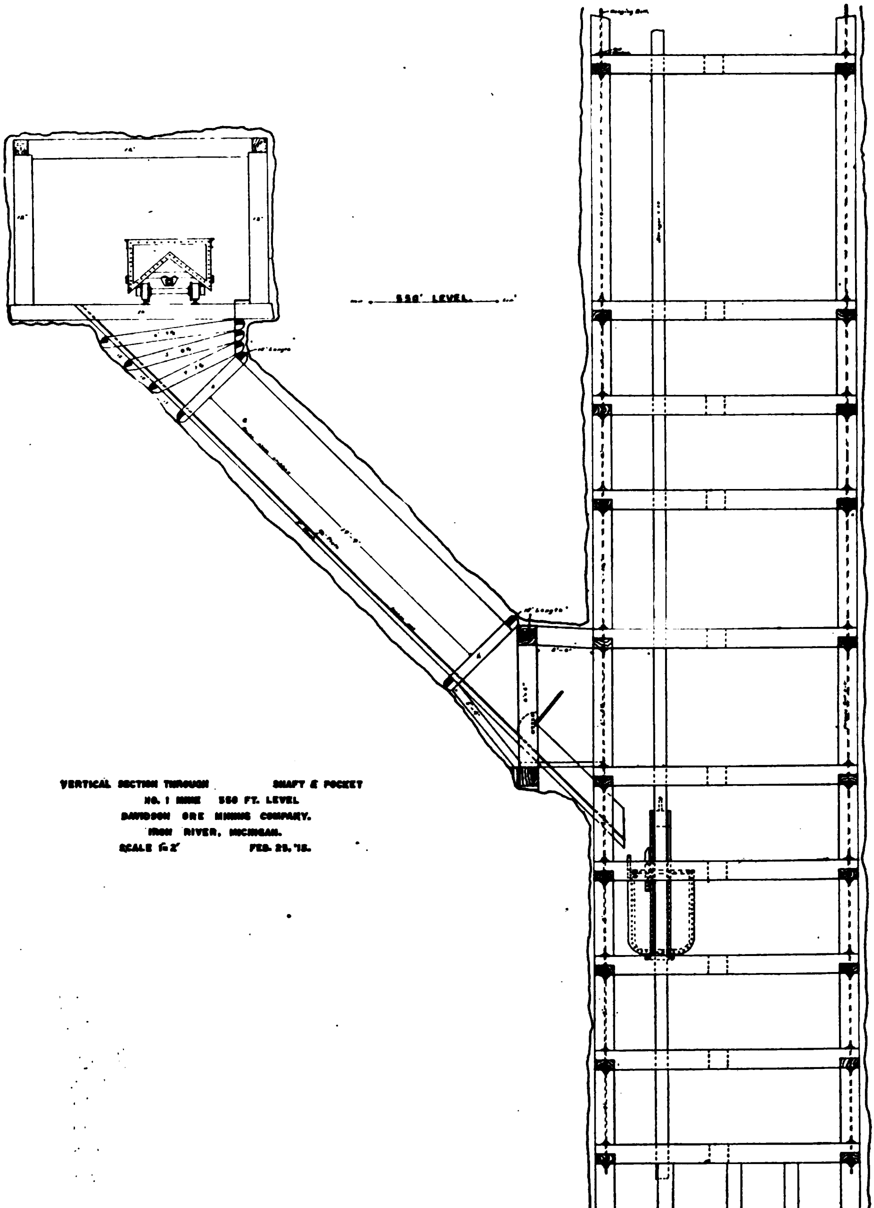
The Davidson No. 2 mine is located about two miles north of the village of Iron River, Michigan, and is operated by the Davidson Ore Mining Company.

Drilling on this property in 1909 and 1910 disclosed a vein of iron ore having a dip to the south of from 30 to 60 degrees and an orebody decidedly irregular, in places attaining a width of more than 100 feet, and in other places being scarcely a drift wide. The major portion of the deposit extends upward to the sand. The footwall is a graphitic slate and the hanging wall a jasper of varying thickness, which is in turn overlaid by slate. The ore is a limonite, sufficiently hard to require the use of power drills for breaking.

The top-slicing or caving system of mining was adopted for the removal of the ore, because the greater portion of the deposit was in direct contact with the sand overburden, and because of the uncertain strength of the hanging wall due to the overlying slate. With the information available at the time mining was started, it seemed reasonably certain that any other system would cause a greater loss of ore in mining than the operators desired.

The shaft was started in August, 1910, and sunk to a depth of 200 feet, the first level being opened at 152 feet. The first ore was hoisted in February, 1911, six months after the clearing of the surface was started. The 152-foot level was opened by drifting in the ore along the footwall contact, and, where the ore was sufficiently wide, driving drifts in the ore on the hanging side. The hanging-wall drifts were merely to improve the ventilation. At intervals of 100 feet on the footwall drift, 60-degree incline raises were put up to within five feet of the top of the ore, and from this point the top or first

*Superintendent Davidson Ore Mining Co., Iron River, Mich.


550' LEVEL
VERTICAL SECTION THROUGH SHAFT & POCKET
NO. 1 MINE 550 FT. LEVEL
DAVIDSON ORE MINING COMPANY.
IRON RIVER, MICHIGAN.

slice was started. Single longitudinal drifts were driven 50 feet each way from the raise, and from the extremities of these drifts, crosscut drifts were driven to the foot and hanging walls. In this way the block of ore to be mined from each raise was outlined. Two crosscuts were next driven parallel to and next to the end crosscut, so as to produce a room 30 feet in width with a length varying as the width of ore in the particular place.

At this stage the ore remaining above the timbers of the slices was mined out. The plan was an endeavor to keep a uniform thickness of 5 feet of ore above the timbers, but the uneven contact between the sand and ore in some places varied this amount greatly. The ore above the top slice in the 30-

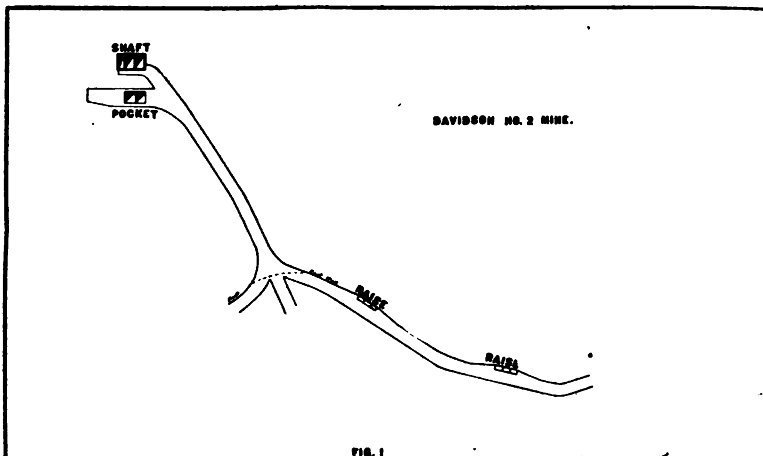


FIG. 1

foot rooms was removed to the sand, usually by a top-timber method. When all the ore had been removed, the bottom was covered, and the sides of the room, excepting on the foot and hanging walls, were laced with plank. The covering of the bottom consisted of 8-foot poles 4 to 6 in. in diameter laid 3 feet apart crosswise of the drift and of 2-in. plank of No. 3 grade laid across these poles and lengthwise of the drift. Particular care was exercised to get all the covering laid before bringing in the sand. In this case directly over the ore was a hardpan that had to be blasted before it started caving. The remaining ore of the top slice was removed in the same way, except that the rooms were not made so large before caving the sand and there was no further need of blasting the hard-pan.

In the lower slices, there is a good matte of timber and cheap 1½-in. maple is used instead of 2-in. plank for lacing the sides and covering the bottom. The side lacing is also sometimes done with a re-sawed material about ⅞-in. in thickness. The regular slice is 11½ feet high. The timber

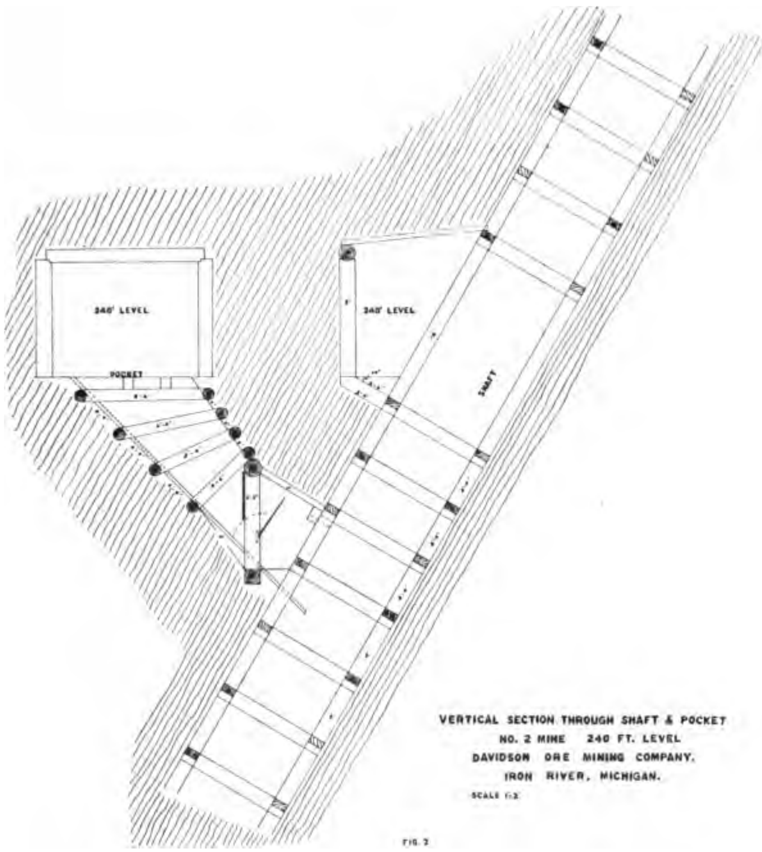


FIG. 2

used is 6½ to 9-in. in diameter and the legs and caps are 8 feet in length. The sets are spaced about 4½ feet apart, depending somewhat on local conditions. No top timber sets are used, the space between the top of the cap and the covering boards of the slice above being filled loosely with a network of lagging. In the regular mining, no ore is left in the back of the slice to be robbed out later, everything being mined clean to the covering boards as the drift progresses.

The raises are cribbed and usually have three compartments, two serving as ore chutes and the center one as a passage-way for workmen and supplies. Each compartment is 4 feet square. A raise of this type is well adapted to the contract system of mining, as two contracts can work from a single raise with separate places for dumping the ore.

The present plan calls for main ore haulage levels at approximately 100-foot intervals. Figure 1, which is self-explanatory, shows the plan of a portion of 240-foot level. The

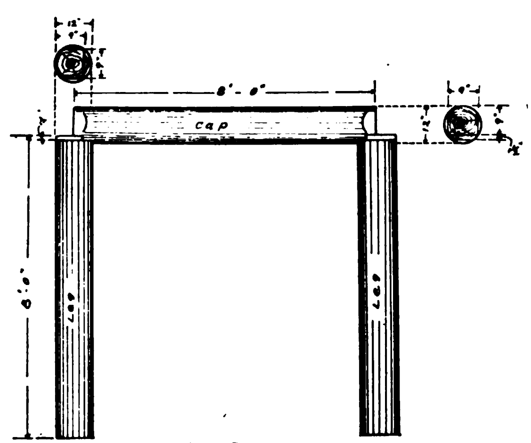


Fig 3
Drift Set Main Level
Davidson N° 2 Mine
Scale ½" = 1'

layout of drifts, and method of installing the pocket at the shaft is further illustrated by cross-section in Figure 2; it is somewhat unusual, but has been found to give satisfaction because it does not cut up the ground and cause it to shatter. The usual method, where a pocket is established at the shaft, of removing all rock material between the pocket and shaft and thus producing a large opening, often becomes troublesome to hold and repair, as close to the shaft troubles occasioned by crushing or moving ground are hardest to combat.

Figure 1 shows the method adopted for opening out the main level at the point where the raise is located. The regular cap used in the main level is 8 feet long, but these are gradually lengthened at raise turnouts to a maximum of 11 feet, and the drift is continued at this width until past the inside ore chute when the cap is again gradually reduced to regular length. There is room at these turnouts for cars to stand under the chutes while another train of cars is passing on the main level. Cars can be pushed or pulled under the chutes from either end, as frogs are placed in the track at each end of the turnout; this system serves to a considerable extent the purpose of a double-track drift without as high an initial cost or as large a maintenance cost.

The cars are hauled by a 4-ton Baldwin-Westinghouse motor of 30-in. gauge. The trolley pole is what is known as D-20 type; it is especially serviceable when the motor is running on the sidetrack at the turnouts, as only one trolley line is used and that line is over the main track, so that the trolley pole has to adjust itself automatically.

The method of framing timber for main levels is shown in Figure 3. This particular method is far from being new and is probably used at other mines. The ease and quickness with which the timber can be framed and set up especially recommends it, also the fact that for openings of a certain size timber is saved, since the cap does not extend over the entire top of the leg. There seems to be ample strength directly over the leg, as the cap breaks in the center. The writer first saw this method of framing round timber at the Lake Angeline mine, at Ishpeming, Mich.

THE SUB-STOPING METHOD OF MINING AS USED AT THE CHATHAM MINE.

BY F. J. SMITH, IRON RIVER, MICH.*

The principal method of mining at the Chatham mine is what is known in this district as sub-stoping. The orebodies are comparatively small, narrow, and irregular, varying in width from 10 to 50 feet and running not usually more than 300 to 400 feet in depth. In many places rock bunches stretch clear across the orebody. The dip varies from 60 to 90 degrees and the ore differs in structure from soft limonite to very hard blue hematite, both often being mixed in the same orebody.

The levels of the Chatham mine are 125 feet apart vertically. On these levels, drifts, and crosscuts if necessary, are driven in the ore in such a manner that raises may be put up about 25 feet apart and chutes built in them. A new raise is started as soon as the drift has advanced sufficient distance from the last one. The raises are extended 25 feet. As soon as the chutes are built, the drift is timbered to prevent any caving of the back when stoping is being done above. At some convenient place in the ore, two of the raises, about 50 feet apart, are continued clear through to the level above. Generally one of these two raises is near the end of the orebody so that stoping may later be started around it. Sub-levels, or "subs" are then put in every 25 feet, vertically, connecting these raises; this makes four "subs" between each level. One of the raises is then ready to be used for ladder- and pipe-way and another for a mill. The subs are then continued toward the end of the orebody. Whenever a breast of the subs gets too far from the original mill for efficient mucking, a raise is put through from the 25-foot sub below. The ore is easy to handle on the 25-foot sub-levels, because, as before stated, the raises are put to this "sub" as fast as drift advances. The subs are continued into the rock a short

*Superintendent of the Brule Mining Co.

distance beyond the end of the orebody and then connected with a raise, called a "blind raise," which is used for ladder-way and pipes and serves as an exit when this end of the orebody is being stoped.

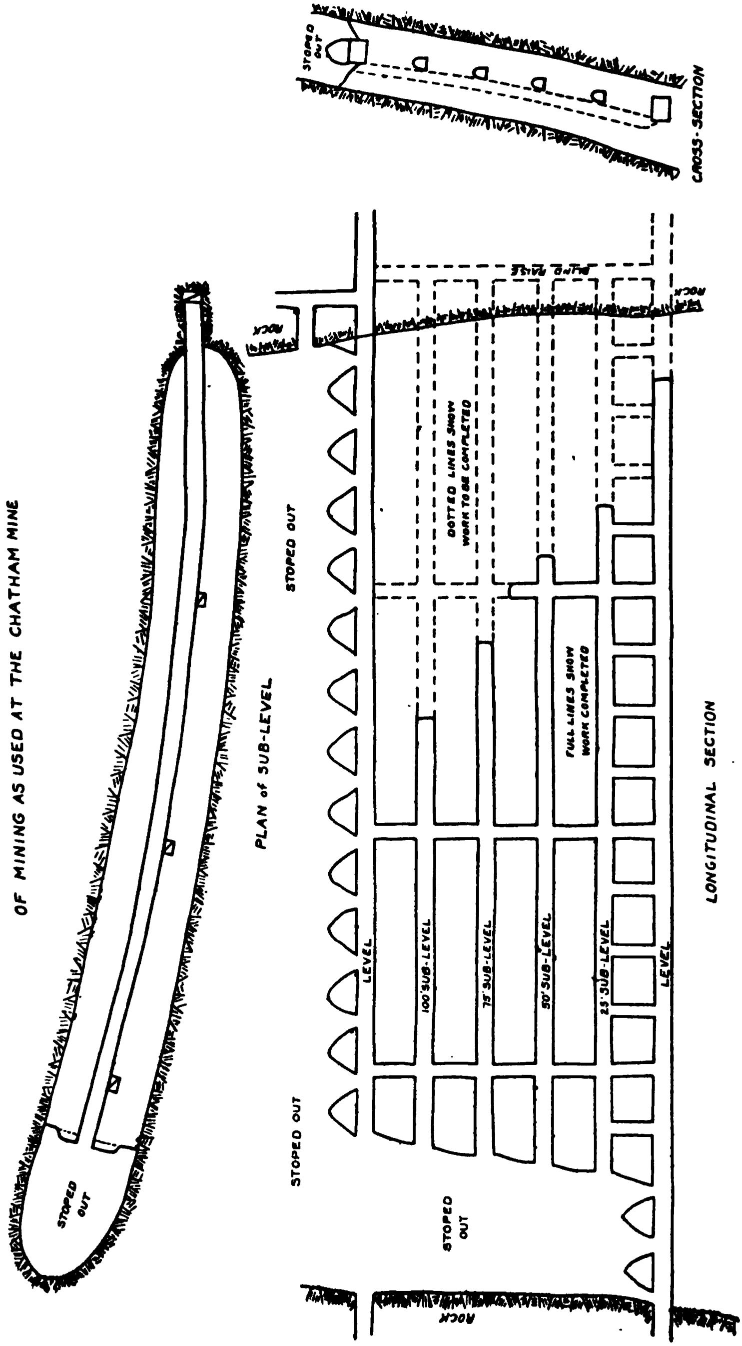
If stoping is started at any point except at the farthest end of the orebody from the blind raise, then a blind raise will be required at both ends of the orebody; therefore, stoping is generally started around the first long raise put up, which is at the end of the orebody. There is an advantage, however, in beginning near the middle of the orebody, in that stoping can then be done towards both ends of the orebody at the same time, twice as many working places being thus provided.

The stope is started by drilling uppers and down holes on each sub around the raise and slicing out the ore toward the foot and hanging until a wide bench is formed, after which more stope holes are drilled. This process is repeated until the full width of the orebody is opened into a stope reaching from the 25-foot sub upward. The level above is worked back at the same time. Particular attention is given to funneling out the raises below the 25-foot sub, so that large chunks of ore can work down close to the mouth of chute where they can be easily blasted. Stoping is continued by slicing out a well-covered bench from the sub toward the foot and hanging. When the rock walls have been reached, uppers and down holes are drilled from the benches, first nearer the rock and then back toward the sub. When this is finished another bench is started from the drift and continued to foot and hanging and the above operation repeated. When a rock bunch is encountered which is large enough to be left for a pillar from foot to hanging, a raise is put up along the rock on the opposite side from the stope and stoping again started in the way previously described. If the rock bunch is smaller and cannot be left it is drilled separately and as soon as the chutes are empty, it is blasted down and hoisted for rock.

Blasting is done as often as possible, so that a great amount of ore does not have to be blasted at one time, for otherwise the ore will run on the 25-foot subs and make the miners there to do a lot of shoveling. Generally 10-foot holes are deep enough, but occasionally 14- and 16-foot holes are needed. The angle of the face of the stope depends upon the character of the ore; if the ore is soft and subject to caving, the stoping on the upper subs is kept a little further advanced than on

SKETCHES SHOWING SUB-STOPING METHOD

OF MINING AS USED AT THE CHATHAM MINE



the next ones below. If the ore stands well, the face is carried about vertical.

Because the orebodies are small, often those located on one level are developed and mined during the same season. Stoping is, therefore, begun before the sub-drifts are finished, as is shown in the sketch of the longitudinal section: the heavy lines indicate entry work completed at the same time that the stope is well started, and the dotted lines show entry work yet to be completed while stoping is still being done.

The following are a few of the advantages and disadvantages of the system:

Advantages.

(1) Little development work is required before stoping may be begun.

(2) All the ground broken can be hoisted at once.

(3) The method may be used to good advantage in ground which is too soft and not uniform enough for back-stoping.

(4) The rock bunches found in the ore are disposed of with little trouble.

Disadvantages.

(1) The large amount of drifting and raising required may make the method uneconomical in hard ore.

MINING METHODS IN THE CRYSTAL FALLS, AMASA AND FLORENCE DISTRICTS.

BY M. E. RICHARDS, CRYSTAL FALLS, MICH.*

The principal mining systems used in the Crystal Falls, Amasa and Florence districts are sub-stoping, back-stoping, block-caving and underhand-stoping. There is one open pit mine in the district. The stoping systems can be used to best advantage because the ore occurs in lenses, ordinarily not wider than 60 feet, with rock capping, in steeply pitching folds in the slate footwall. The backs are thus narrow, if properly arched, and strong enough to hold up the surface while the lens is being mined.

In some instances, when the ore outcrops at the surface or when the rock is not thick enough to support the surface, it is necessary to leave a small arch of ore. In other cases when there is danger of the hanging caving into the stope, it is necessary to leave a small pillar of ore from foot to hanging. The major portion of these pillars can be mined out after the rest of the ore is mined from the lens.

When the ore extends to greater depths, at times it is important to hold a hanging from coming in, in which case the back-stoping method of mining is usually adopted. By this method, the ore is broken, working each time from the lower level up to the next higher, only enough ore being drawn off to make working room for the miners. When the last slice of ore up against the broken rock covering is reached, it is first broken through to the loose rock at the far end of the stoping, after which the miners retreat, breaking the ore through as they go, toward the traveling ways. The loose rock now lies on top of the broken ore, and as this broken ore is drawn off from the chutes, the rock capping gradually settles, following the ore down to the lower level. The loose rock following the ore down is usually not barren, but runs from 40 to 49 per cent in iron; therefore if a small quantity

*General Manager, The Judson Mining Co., Alpha, Mich.

is mixed with the ore, no serious damage results. This method of stoping is used to some extent at the Judson mine, and almost entirely at the Bristol mine. It will be described in detail by the Bristol Mining Company in an accompanying paper. The two main advantages of this system of mining are that by means of it a caving back can be handled and a minimum amount of development work is required.

The sub-stoping method is used most of all in this district and will be described in detail by the Judson Mining Company in an accompanying paper.

The block-caving system is used in some mines where the lens of ore is extremely wide, so that the swing would be too great for the back to stand, if either back-stoping or sub-stoping were used; or in other mines where the ore is not hard enough to hold up with a large back, and large chunks continually drop away, making the working places unsafe for the miners. Suppose, for example, that a block of ore 150 feet long by 150 feet wide and 100 feet between levels is to be mined by the block-caving method: First, raises 15 feet apart are put up from the bottom level, to a height of 20 feet in the pillar of ore. At the top of these raises a sub-level is opened, cutting across to each raise. When the bottom of the block of ore is thus honey-combed, all the small pillars on this sub are thoroughly drilled, and then all are blasted at once. This under-cuts the block of ore, so that it gradually settles and can be blasted up as it settles into the chutes. This method of mining was described in detail by the Corrigan McKinney Company in its account of methods of working the Tobin mine in 1911. (Vol. XVI.)

The underhand stoping method is used in some mines and will be described in detail by the Florence Mining Company in an accompanying paper.

Open-pit mining is practiced at the Balkan mine and was described in detail by the Balkan Mining Company in 1916. (Vol. XX.)

SUB-STOPING AT THE AMASA-PORTER MINE.

BY M. E. RICHARDS, CRYSTAL FALLS, MICH.*

The Amasa-Porter mine is located about two and a half miles southeast of Amasa on the east ½ of the northeast ¼ of section 22-44-33, Michigan. It is operated by the Judson Mining company.

STRUCTURAL GEOLOGY.

The orebody lies mostly in the syncline of a broad fold between layers of ferruginous slate, the footwall of which is black slate and the hanging wall chert and jasper. The orebody averages 35 feet wide and the formation dips at an angle of 80 degrees. The ore does not outcrop at the ledge, but blends into the ferruginous slate to form a rock capping 80 feet thick over the orebody.

MINING METHOD.

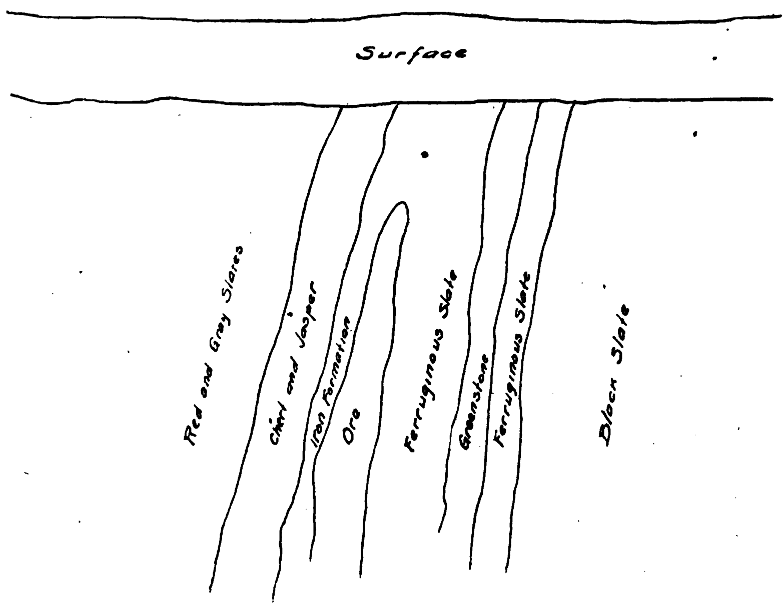
The method of mining used at the Amasa-Porter mine is sub-stoping. This method was devised for a long deep orebody not over 50 or 60 feet wide and dipping at an angle of 60 to 90 degrees from horizontal. The ore should not extend to surface, and the rock capping over the ore must be strong enough to support the overburden, the thickness of it varying with the width of the orebody and the character of the rock. The ore itself must be of such structure that it will stand in the stopes and sub-drifts without the aid of timber. The hanging, if of rock, must be solid and able to stand unsupported; if of rich iron formation, it must be rathed hard. However, should a slab of this formation fall and mix with the ore in the stope, the grade of the product will not be seriously affected.

The Amasa-Porter orebody seemed to have all the main requirements for sub-stoping—firm structure, a rich iron formation for a hanging, and a long narrow lens of good depth

*General Manager, The Judson Mining Co., Alpha, Mich.

and with a strong back. On the main or tramming levels, which are 150 feet apart, is the only place where it is necessary to use timbers, the timbers here being mainly to hold the chutes.

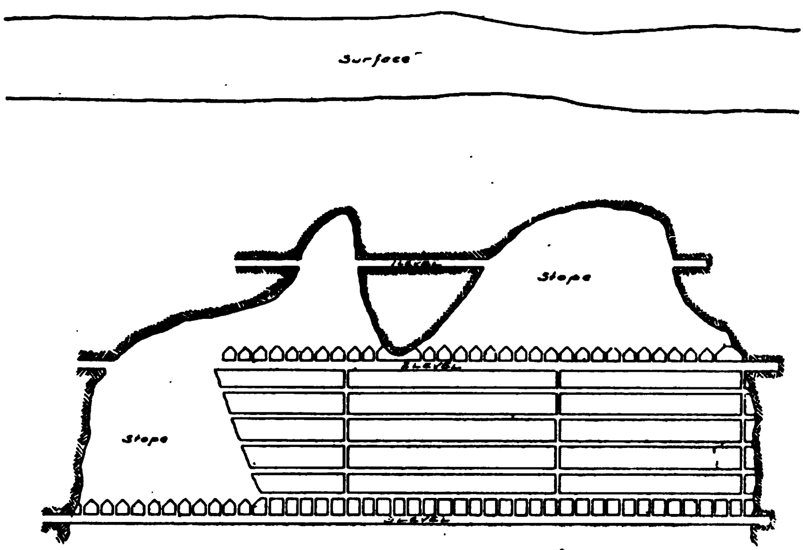
The main drifts from the shaft are driven in rock to the ore, thence in ore along the footwall to the ends of the orebody. Several crosscuts are driven to determine width. At the extreme ends of the drifts and at one advantageous point



Amasa Porter Mine
Typical Cross Section
Showing
Structural Geology

near the center of the orebody, raises are extended to the next level. The end raises are places to start stopes and the center raise serves as a traveling way. The first sub is started from a traveling road, its bottom being 15 feet above the back of the main level. It follows the middle of the ore through to the long raises at the extremities of the orebody. The second and third subs are opened in the same way, except that all the subs after the first are twenty feet apart instead of fifteen, The distance between subs varies somewhat depending upon the way the ore breaks. Intermediate raises are then put in, 15 feet apart, center to center, from the main level up to the first sub. These serve as chutes.

When the development work is completed, stoping is started on the first sub around the long raises at the ends of the orebody. Upper and underhand holes are drilled and blasted around the raise, the ore falling into the chute at the bottom.

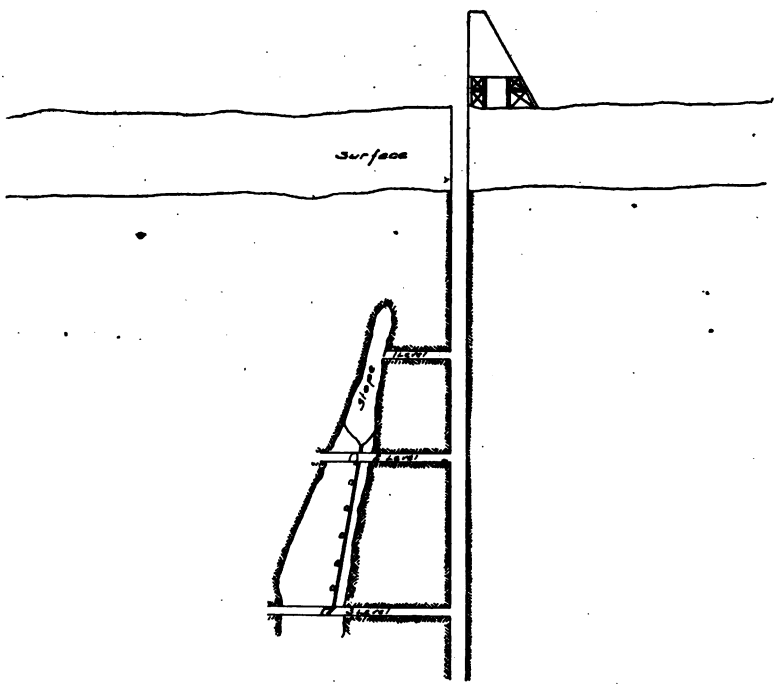


Amasa Porter Mine
Longitudinal Section
of
Sub Stoping
Scale

This is continued until the stope extends from foot to hanging and back 12 or 15 feet from the raise. The second sub is then worked in a like manner, and also the third, as soon as the stope below has been drawn back a safe distance. Thus the men are always working under a solid back, as the ore which is broken on the upper subs falls clear of the miners on the subs below.

In stoping, the ore is taken out absolutely clean, as the benches on each sub are always carried across the orebody from foot to hanging, so that no ore is left unmined if reasonable precaution be taken to show the rock at the ends of each bench.

Once a mine is well started upon this system, the largest part of the work is tramming, for a couple of gangs of miners can keep a large crew of trammers working. It is advisable to have the main level drifts double-tracked so as not to cramp



Amasa Porter Mine
Cross Section
of
Sub Stoping
Scale

the trammers, and so that in case a chute overflows and blocks one track, way is still open to the shaft on the other. Switches should be placed at regular intervals. The main difficulty, however, is in getting the ore from the chutes, for it often comes from the stopes in large chunks, which must be blasted before they can pass the chutes.

MINING METHODS IN THE FLORENCE DISTRICT.

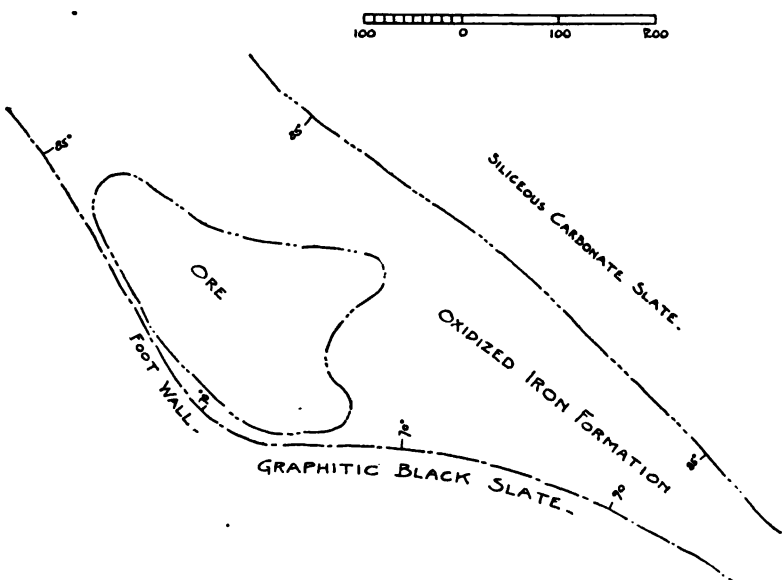
BY J. M. RIDDELL, FLORENCE, WIS.*

At the present there are but two active mines in the Florence district, namely, the Florence mine, at Florence, and the Ernst mine, at Commonwealth. The Florence mine is the oldest mine in the county, but for the past four years it has not been as active as the Ernst mine.

ERNST MINE—STRUCTURAL GEOGLOGY.

The orebody rests upon a footwall of black graphitic slate,

FIG. 1.— ERNST MINE



which dips about 70 degrees from the horizontal. Figure 1, a horizontal projection, shows that the footwall varies in

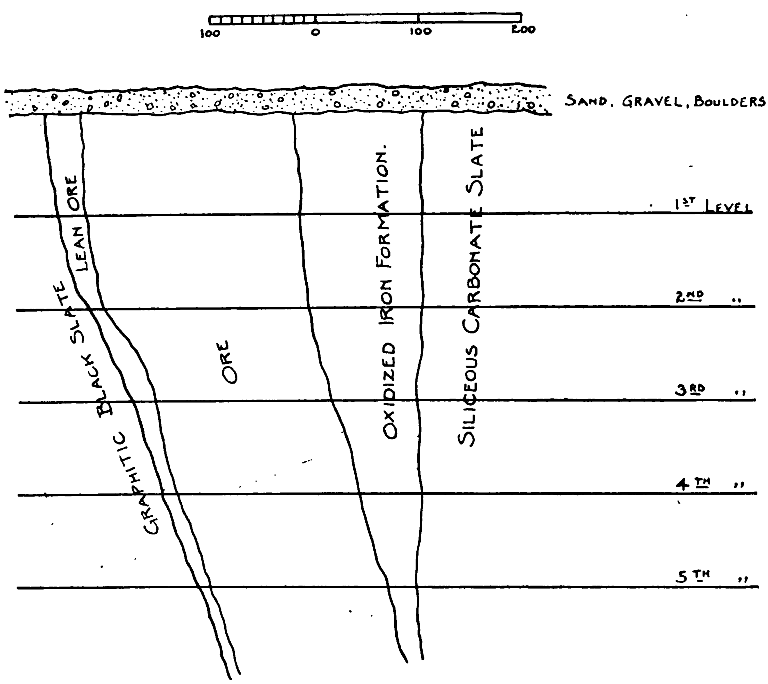
*Mining Engineer, Florence Mining Co., Florence, Wis.

direction; the orebody, it will be noted, adapts itself closely to the distortion of the footwall. Resting upon the orebody is an oxidized iron formation, to the northeast of which is a hard silicious carbonate slate, which forms the hanging wall. Figure 2 shows a typical cross-section.

MINING METHODS.

Because of the firmness of the ore and the manner in which the "back" would stand, it was deemed advisable to adopt what is commonly known as the "underhand-stoping method" or "underhand-caving method." This method gave

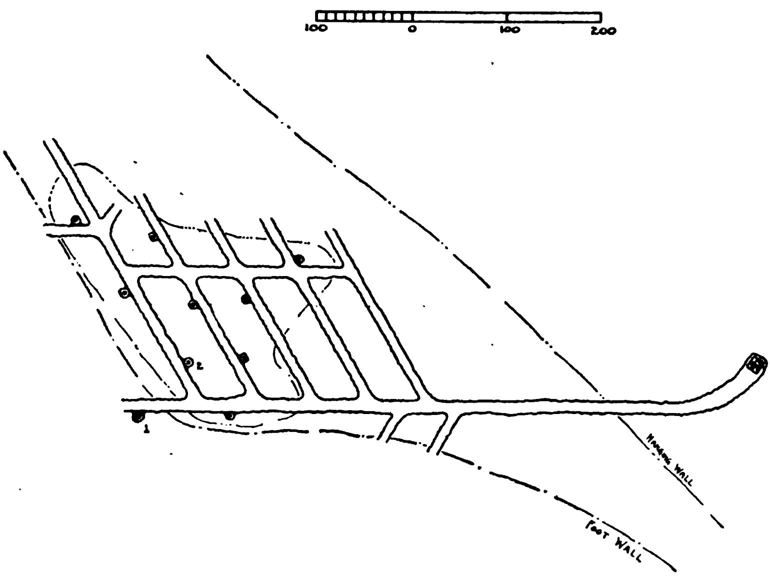
FIG. 2 – ERNST MINE



splendid results between the ledge and the third level, but between the third and fourth levels, sub-stoping had to be combined with it because the back became more friable. From the collar of the shaft to the first level is a distance of 120 feet; all the other levels are 90 feet apart. Figure 3 shows the general scheme adopted in developing and laying out the levels. The main haulage way is close to the footwall. The crosscuts, which make an angle of 60 degrees with the main haulage way, are 60 feet apart. The auxiliary drifts are located according to the shape of the orebody as revealed by the crosscuts. Raise No. 1 is driven from level to level in the footwall and connects with all the sub-levels, thus providing ventilation and affording a secondary passageway to surface. Raise No. 2, located in the orebody, connects with the level above and serves as a chute in the development of the sub-level, which is within 25 feet of the level above.

The sub-level drifts and crosscuts are all driven in ore and

FIG 3 - ERNST MINE



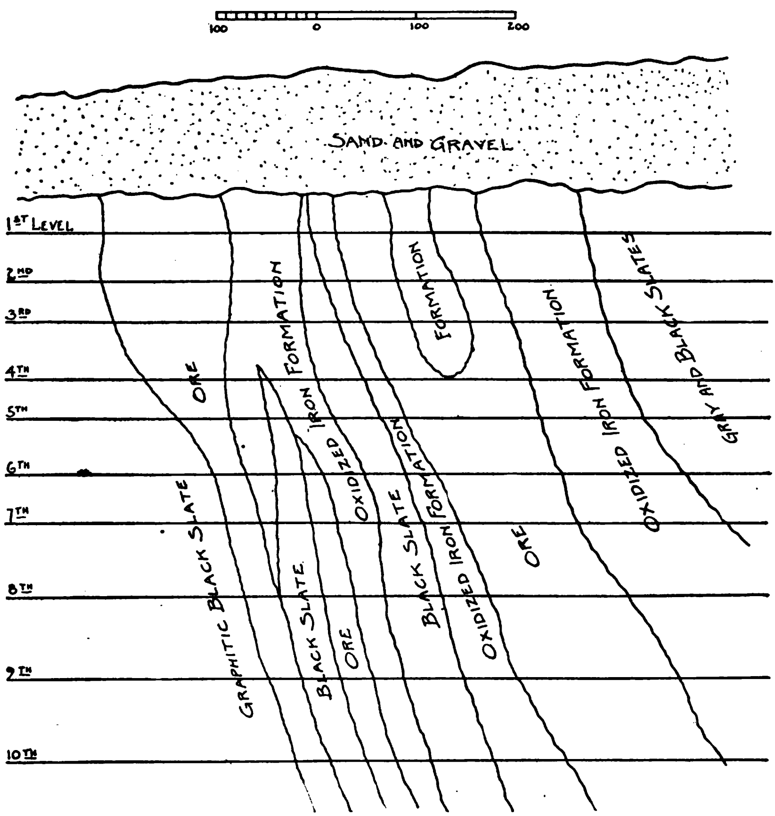
in direct relation to the workings on the lower level. This arrangement of the drifts and crosscuts between level and sub-level makes the extracting of the ore very simple. After the sub-level is developed, numerous raises are driven about 60 feet apart, center to center, between the level and the sub-level above. By underhand stoping methods, these raises terminating on the sub-level are then funneled out until the angle of repose is reached, when the ore will no longer discharge into the chutes. Additional raises are then driven from the lower level, which in turn are funneled out, and this method is continued until the ore is all extracted; this requires raises spaced about 30 feet center to center.

When the stoping is completed, a portion of the floor pillar on the upper level is removed by overhand and underhand stoping. However, a pillar about 20 feet wide and 25 feet thick is allowed to remain to give support to the hanging wall and thus prevent caving.

FLORENCE MINE—STRUCTURAL GEOLOGY.

Figure 4 is a typical cross-section of the orebody and its

FIG. 4—FLORENCE MINE



association. The footwall is a graphitic black slate, upon which usually lies a triplicate series of iron ore, oxidized iron formation, and black slate. This condition prevails near the center of the mine, but at either end the center series pinches out and leaves two series which are nearly parallel. The ore-body varies in width from 10 to 130 feet, and its length is approximately 2,500 feet.

Mining Methods.

In the twenty-two years between 1878, when the mine was first opened, and 1900, a number of different methods were employed in extracting the ore: Considerable of it was mined from an open pit; underground, the common method was breast-stoping. However, the greater part of the ore has been removed by underhand stoping. The method is very similar to that employed at the Ernst mine. To keep the walls from caving, and make possible the removal of numerous supporting pillars, which were in ore, a number of the abandoned stopes were filled with sand, this material being accessible because it originally capped the deposit.

HERRINGBONE GEARS USED ON PUMPS.

BY FRED M. PRESCOTT.*

We are often asked our reason for using herringbone gears on our electric- and power-driven pumps, and why we consider these gears superior to the older forms of straight-tooth or spur gears. Some enlightenment on this subject is due and no doubt a few words regarding their installation and care will be read with interest by all pump users.

The work of the pinion gear is to so drive the other gear that it will revolve steadily and continuously; consequently there should be no intermittancy in the tooth pressure at any time or in any part of the revolution. Spur gears do not fully answer this requirement, while herringbone gears do. The method by which a small spur pinion drives a larger gear has been described as striking its teeth a succession of blows with a hammer. This may sound like an exaggerated statement, yet it describes precisely the action of a straight-toothed pinion, especially if it has but few teeth or becomes worn.

Herringbone gears are really double helical gears, with the helix (spiral angle) of the teeth sufficiently steep to insure an overlapping or screwlike contact between the adjacent teeth, thus compelling a uniform, even and continuous shove or push in the direction of the load and throughout the entire revolution. It is clear, therefore, that this form of gear much more nearly approaches the ideal in the performance of the work desired.

The fundamental and conspicuous advantages of the double helical gear are its greatly increased mechanical efficiency and the high ratios of reduction permissible between gear and pinion. These advantages are of the greatest value to users of geared pumps because they simplify construction and lengthen the life and increase the efficiency of the entire installation.

It must be remembered, however, that to secure these values, gears of this kind must be cut with a precision and ac-

*President of The Prescott Company, Menominee, Mich.

curacy that is hardly believable. Only the most delicately adjusted machines can be used, coupled with workmanship of the highest order, producing a degree of mechanical exactness through the whole process that is seldom fully appreciated even by those who buy the gears. It follows then that the greatest care and attention must also be given to installing such gearing and this work should be intrusted only to an

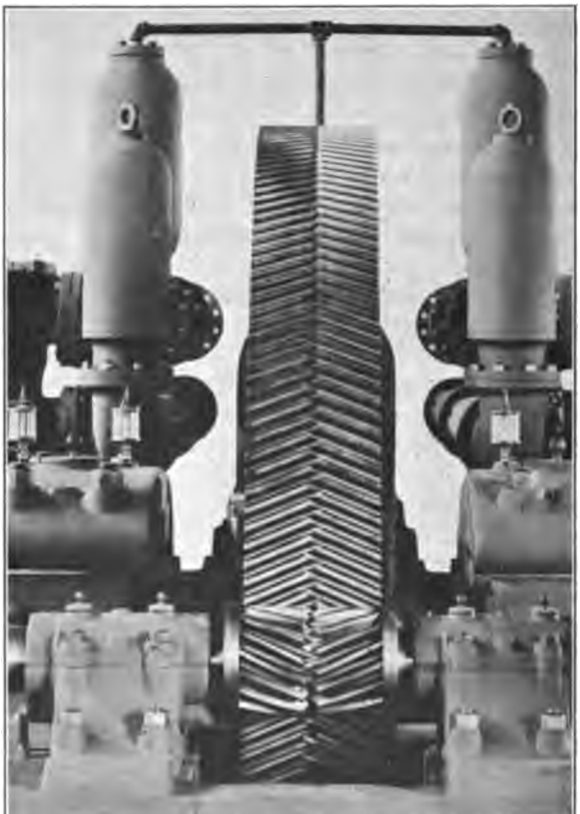


FIGURE 1. SHOWING TYPE OF HERRINGBONE GEAR REFERRED TO IN THIS PAPER.

experienced and skilled mechanic. Experience has shown that nearly all gear troubles have been because of faulty alignment at the time of erecting and starting the machines. A word of caution, amounting to almost a warning, cannot be sounded too loudly on this phase of the subject, so vital is it to the welfare, life and efficiency of any geared pumping unit.

Because the pumps have been erected, and the gears leveled, properly lined up and adjusted in the shop, it does not follow that after the installation has been knocked down, shipped and re-erected on some other foundation, everything will go together again accurately enough to run properly. The utmost care must again be taken to check every level and alignment and to be sure on the following points:

1. That the pump is erected on a firm and solid foundation.
2. That the frames are set absolutely level, the proper distance apart and parallel.
3. That the levels of the frames are maintained after they and other parts of the pump have been grouted in and the foundation bolts have been pulled up tight. (This is to be sure that the frames are set without any undue strains.)
4. That the gear shaft is absolutely level.
5. That the pinion shaft is level.
6. That the gear shaft and pinion shaft are absolutely parallel.
7. That the gear and pinion teeth mesh with approximately 0.015 clearance or back lash.
8. That there are no dirt and chips between the gear hub and the shaft, and that the joints and hub are clean, and that all of the bolts are brought home tight.
9. That the hub has a good bearing on the shaft all around and that it does not ride on the top of the key when tightened.
10. That the pinion shaft has a good bearing the entire length of its boxes—a dummy shaft should be used to ascertain this, and the bearings scraped to the required exactness.
11. That the motor shaft is in exact line with the pinion shaft.
12. That the halves of the flexible coupling are set sufficiently apart to allow for end motion.
13. That the flexible parts in the coupling have ample freedom to allow for extensibility as well as flexibility.
14. That the gears are never without lubrication.
15. That the main bearings are pulled up fairly snug, so as to eliminate any side motion of the gear.
16. That, should the main bearings ever become exces-

sively worn, shims be placed back of the pillow block quarter boxes so as to keep the gears at their original centers.

It must above all be impressed on the mechanic that straight edges, levels and other tools of approximate correctness will not do, for with gearing whose teeth are generated to register perfectly, the settings must be exact to the thousandth part of an inch. If so maintained, gears of this kind will last a long time; but if any of these things is not carefully looked after at the very outset, trouble can be expected later on, for it will surely come.

There is a prevailing idea that herringbone teeth are silent, but this is not true. It cannot be expected that gearing of any kind having metal-to-metal teeth and transmitting heavy loads will be entirely noiseless. However, they are much more quiet than other forms of cut gears, and, when all of the precautions enumerated above are complied with, noise is practically eliminated and what little remains is never objectionable.

Herringbone gears should be lubricated freely and constantly. With slow running gears doing heavy work, as in mine pumps, the lubricant should be heavy, viscous, and above all things, free from grit. When first starting up the gears, a good thing to do, after cleaning them thoroughly, is to slush the teeth freely with white lead, and then run them that way under full load long enough to insure against seizing. This protects them from the common danger of biting or cutting when picking up their initial load, which, once it has happened, is the sure forerunner of serious wear. After that, clean and dry the gears again thoroughly and apply the regular lubricant.

In all cases where it is possible, geared pumps should be erected under the supervision of a man sent from the factory, but where this it not feasible and due regard is given to the essential accuracies pointed out in this article, the pump will run so smoothly, efficiently and quietly that it will be a joy to behold and a pleasure to own.

MINE ACCIDENTS CLASSIFIED BY MINING METHODS, FOR THE LAKE SUPERIOR DISTRICT, 1915.

BY ALBERT H. FAY, WASHINGTON, D. C.*

As a result of the work of the United States Bureau of Mines in collecting statistics relating to mine accidents, I am able to present herewith the accident data for the Lake Superior district, classified by causes and mining methods, for the year 1915. The data here given represent the reports of 116 iron, copper, and lead-and-zinc properties in Michigan, Wisconsin, and Minnesota. By methods, these properties are classified as follows: overhand stoping with shrinkage, 2; overhand stoping with filling, 9; overhand stoping with the use of square sets, cribs, and stulls, 13 copper and 16 iron properties—the copper mines do not appear in any other group; caving system, 49 properties all iron; room-and-pillar method, 3 lead-and-zinc mines and 7 iron mines; steam shovel operations, 16 iron properties in Michigan and Minnesota.

The properties have been classified according to the principal method used, which in the majority of cases represents 80 or 90 per cent of the total work done. Any mine may use overhand stoping as its principal method, but may also do a certain amount of underhand stoping, a small amount of filling, and even some room-and-pillar work. The figures given are compiled from selected reports of companies whose records seemed reasonably complete, all incomplete returns being thrown out.

The total number of men reported employed in the mines of the Lake district for these properties was 29,883, as compared with 41,154 tabulated by mining methods for the remainder of the United States. The figures selected include the actual number of men employed underground and in open-pit mines with the corresponding number of accidents for the group. The employes around shops and surface workings have

*Mining Engineer, Bureau of Mines.

been eliminated from these numbers, as well as the corresponding number of accidents.

Table 1 shows the actual number of men employed in each of the various groups of mines in the Lake district as compared with "Other United States." It also shows the number of men killed and injured, together with the rate per 1,000 employed. It also shows the average number of days worked per year, which in the Lake district was 288 as compared with 323 for the remaining states. The group of mines showing the largest number of days worked in the Lake region is the

TABLE 1. MINE ACCIDENTS IN LAKE SUPERIOR REGION COMPARED WITH OTHER STATES BY MINING METHODS AND ACTUAL NUMBER OF EMPLOYES. (1915)

	Over hand Stoping			Caving	Room and Pillar	Steam Shovel	Total
	Shrinkage	Filling	With Timber				
Lake Superior Region:							
Number of employes:	239	3086	7389	14425	787	3957	29883
Number killed	1	15	37	50	0	12	115
Number killed per 1,000	4.18	4.86	5.01	3.47	....	3.03	3.85
Number injured (a)	82	1033	3431	3733	307	609	9195
Number injured per 1,000	343.10	334.74	464.34	258.79	390.09	153.90	307.70
Average days worked per year	304	295	296	286	278	273	288
Other United States:							
Number of employes:	4724	17218	5085	4127	6630	3370	41154
Number killed	35	84	20	17	27	12	195
Number killed per 1,000	7.41	4.87	3.93	4.12	4.07	3 56	4.74
Number injured (a)	2162	6621	2339	2068	2512	1299	17001
Number injured per 1,000	457.66	384.54	459.98	501.09	378.88	385.46	413.11
Average days worked per year	349	342	309	286	276	351	323

(a) Includes all injuries resulting in a disability of more than one day.

one using shrinkage stoping, with 304 days as compared with 349 days for the same group of mines in other states. The steam-shovel mines were operated 273 days as compared with 351 days in other states.

Table 2 represents the same group of mines with all of the accident rates equated to a 300 day basis, both for the Lake district and for "Other United States." Inasmuch as the workman on the average was exposed to accident for only 288 days in the Lake district, and for 323 days in the other states, accident rates can not be fairly compared on the basis of the actual number of men employed.

Table 3 shows fatalities and injuries classified by causes for the mining-method groups, both for the Lake district and for the remainder of the United States. The Lake district reports 115 fatalities and 9,195 injuries as compared with 195 fatalities and 17,001 injuries for the other states. The fatality rate (Table 2) on the 300-day basis for the Lake district is 4.01 per 1,000 as compared with 4.39 for "Other United States." The number injured per thousand 300-day workers

TABLE 2. MINE ACCIDENTS IN LAKE SUPERIOR REGION COMPARED WITH OTHER STATES BY MINING METHODS AND ON A 300-DAY BASIS. (1915)

	Over-hand stoping			Caving	Room and Pillar	Steam Shovel	Total
	Shrinkage	Filling	With Timber				
Lake Superior Region:							
300-day workers	242	3034	7301	13734	716	3663	28690
Number killed	1	15	37	50	0	12	115
Number killed per 1,000	4.13	4.94	5 07	3.64	....	3.28	4.01
Number injured	82	1033	3431	3733	307	609	9195
Number injured per 1,000	338.84	340.47	469.94	271.81	428.77	166.26	320.49
Other United States:							
300-day Workers	5499	19622	5240	3945	6094	3945	44345
Number killed	35	84	20	17	27	12	195
Number killed per 1,000	6.37	4.28	3 82	4 31	4.43	3.04	4.39
Number injured	2162	6621	2339	2068	2512	1299	17001
Number injured per 1,000	393.16	337.43	446.37	524.21	412.21	329.28	383.38

in the Lake district was 320.49 as compared with 383.38 for the remaining states.

SUMMARY FOR THE FIVE-YEAR PERIOD, 1911-1915.

Table 4 shows the total number of men killed, by causes, in the States of Minnesota, Michigan and Wisconsin for the years 1911 to 1915 inclusive. This table also shows the percentage of fatalities due to each cause for the four groups of accidents named: underground, shaft, surface and open pit. It will be noted that of the underground fatalities falls of rock are responsible for 59.14 per cent, while of the open pit fatalities 30.17 per cent are due to the same cause. Of the underground fatalities 8.95 per cent are due to explosives, while in

TABLE 3. NUMBER OF MEN KILLED AND INJURED IN THE LAKE SUPERIOR MINES IN 1915, CLASSIFIED BY CAUSES AND MINING METHODS.

	Over-hand Stoping						Caving		Room and Pillar		Steam Shovel		TOTAL	
	Shrinkage		Filling		With Timber									
	Fatal	Nonfatal	Fatal	Nonfatal	Fatal	Nonfatal	Fatal	Nonfatal	Fatal	Nonfatal	Fatal	Nonfatal	Fatal	Nonfatal
UNDERGROUND														
NUMBER KILLED OR INJURED BY—														
1. Fall of rock or ore from roof or wall	....	3	8	305	24	729	28	1059	....	56	....	...	60	2152
2. Rock or ore while loading at working face or chute	....	7	....	157	....	1050	1	328	....	61	....	....	1	1603
3. Timber or hand tools	....	3	....	89	1	25[illegible]	....	742	....	17	....	....	1	1109
4. Explosives	...	....	2	4	2	40	4	38	....	7	....	...	8	89
5. Haulage system (mine cars, mine locomotives, breakage of rope, etc)	....	23	1	117	....	471	1	473	....	30	....	....	2	1114
6. Falling down chute, winze, raise, or stope	....	2	2	34	2	35	6	92	....	1	....	....	10	164
7. Run of ore from chute or pocket	....	10	1	42	1	181	3	61	....	24	....	....	5	318
8. Drilling accidents (by machine or hand drills)	....	2	....	70	....	191	....	146	....	31	....	....	....	440
9. Electricity	....	....	1	20	...	4	1	10	....	....	....	..	2	34
10. Machinery (other than locomotives or drills)	....	....	....	....	....	17	....	22	....	2	....	....	....	41
11. Mine fires	....	....	....	....	....	3	....	1	....	...	....	....	....	4
12. Suffocation from natural gases	....	....	....	....	....	5	1	10	...	....	....	....	1	15
13. Inrush of water	....	....	....	..	....	....	....	....	....	...	....	....	....	....
14. Nails, splinters, etc	....	1	....	62	....	10	....	88	....	5	...	....	....	166
15. Other causes	....	29	....	102	....	337	1	508	....	64	....	...	1	1040
Total number killed or injured underground	....	80	15	1002	30	3331	46	3578	0	298	0	0	91	8289
SHAFT														
NUMBER KILLED OR INJURED BY—														
16. Falling down shafts	....	1	....	4	2	11	1	6	....	....	..	....	8	22
17. Objects falling down shafts	....	....	....	6	1	44	1	50	....	1	....	....	2	101
18. Breaking of cables	....	....	....	....	....	....	....	1	....	....	...	....	....	1
19. Overwinding	....	....	....	....	....	....	....	....	....	....	...	....	....	....
20. Skip, cage or bucket	1	1	....	12	3	7	1	47	....	3	....	....	5	70
21. Other causes	....	....	....	9	1	38	1	51	....	[illegible]	....	....	2	103
Total number killed or injured by shaft accidents	1	2	....	31	7	100	4	155	0	9	0	0	12	297
OPEN-PIT														
NUMBER KILLED OR INJURED BY—														
31. Fall of slides of rock or ore	....	....	....	....	....	....	....	....	....	....	1	121	1	121
32. Explosives	....	....	....	....	....	....	....	....	....	....	1	14	1	14
33. Haulage system (cars, locomotives, etc.)	....	....	....	....	....	....	....	....	....	....	8	78	8	78
34. Steam Shovels	....	....	....	....	....	....	....	....	....	....	1	32	1	32
35. Falls of persons	....	....	....	....	....	....	....	....	....	....	....	61	....	61
36. Falls of derricks, boom, etc	....	....	....	....	....	....	....	....	....	....	....	2	....	2
37. Run or fall of ore in or from ore bins	....	....	....	....	....	....	....	....	....	....	....	6	....	6
38. Machinery (other than locomotives or steam shovels)	....	....	....	....	....	....	....	....	....	....	....	3	....	3
39. Electricity	....	....	....	....	....	....	....	....	....	....	...	....	....	....
40. Hand tools	....	....	....	....	....	....	....	....	....	....	....	97	....	97
41. Other causes	....	....	....	...	....	....	....	....	....	....	1	195	1	195
Total number killed or injured by open-pit accidents											12	609	12	609
GRAND TOTAL	1	82	15	1033	37	3431	50	3733	0	307	12	609	115	9195

TABLE 4. NUMBER OF MEN KILLED IN THE LAKE SUPERIOR MINES, 1911 TO 1915, INCLUSIVE.

UNDERGROUND	MINN.	MICH.	WISC.	TOTAL	PER CENT
NUMBER KILLED OR INJURED BY—					
1. Fall of rock or ore from roof or wall	56	222	26	304	59.14
2. Rock or ore while loading at working face or chute	5	6	....	11	2.14
3. Timber or hand tools	6	5	....	11	2.14
4. Explosives	9	32	5	46	8.95
5. Haulage system (mine cars, mine locomotives, breakage of rope, etc)	13	17	1	31	6.03
6. Falling down chute, winze, raise, or stope	10	34	2	46	8.95
7. Run of ore from chute or pocket	....	19	....	19	3.69
8. Drilling accidents (by machine or hand drills)	....	....	....	....	....
9. Electricity	....	5	1	6	1.17
10. Machinery (other than locomotives or drills)	2	3	1	6	1.17
11. Mine fires	....	10	....	10	1.95
12. Suffocation from natural gases	....	2	....	2	.39
13. Inrush of water	3	8	2	13	2.53
14. Nails, Splinters, etc	....	....	....	....	....
15. Other causes	5	4	....	9	1.75
Total number killed or injured underground	109	367	38	514	100.00
SHAFT ACCIDENTS					
NUMBER KILLED OR INJURED BY—					
16. Falling down shafts	16	42	3	61	40.67
17. Objects falling down shafts	4	13	2	19	12.67
18. Breaking of cables	....	4	....	4	2.67
19. Overwinding	....	1	....	1	.66
20. Skips, Cage, or bucket	5	35	2	42	28.00
21. Other causes	8	10	5	23	15.33
Total number killed or injured by shaft accidents	33	105	12	150	100.00
SURFACE ACCIDENTS					
AT SURFACE YARDS AND SHOPS					
NUMBER KILLED OR INJURED BY—					
22. Mine cars or mine locomotives, gravity or aerial trams	....	4	....	4	7.55
23. Railway cars and locomotives	2	6	....	8	15.09
24. Run or fall of ore in or from ore bins	3	2	2	7	13.21
25. Falls of persons	3	6	1	10	18.87
26. Nails, Splinters, etc	....	....	....	....	....
27. Hand tools, axes, bars, etc	....	....	....	....	....
28. Electricity	....	2	....	2	3.77
29. Machinery	1	3	1	5	9.43
30. Other causes	2	14	1	17	32.08
Total number killed or injured by surface accidents	11	37	5	53	100.00
OPEN-PIT ACCIDENTS.					
NUMBER KILLED OR INJURED IN PIT BY—					
31. Falls or slides of rock or ore	34	1	....	35	30.17
32. Explosives	16	....	....	16	13.79
33. Haulage system (cars, locomotives, etc)	47	....	....	47	40.52
34. Steam shovels	5	....	....	5	4.31
35. Falls of persons	4	....	....	4	3.45
36. Falls of derricks, booms, etc	3	....	....	3	2.59
37. Run or fall of ore in or from ore bins	....	....	....	....	....
38. Machinery (other than locomotives or steam shovels	2	....	....	2	1.72
39. Electricity	....	....	....	....	....
40. Hand tools	....	....	....	....	....
41. Other causes	4	....	....	4	3.45
Total number killed or injured by open-pit accidents	115	1	....	116	100.00
Grand total	268	510	55	833	

open-pit mines 13.79 per cent are due to this cause. Haulage systems are responsible for 6.03 per cent of the fatalities underground, and 40.52 per cent of the open-pit fatalities.

Table 5 shows labor and accident data for the Lake Superior district for 1911 to 1915 inclusive, giving by year and by states the number of men employed and days active, the number killed and injured, and the rates per 1,000 men employed.

The figures given in Tables 4 and 5 give all of the fatalities that were reported to the Bureau and are assumed to represent the complete mining industry of each state. The figures in Tables 1 to 3 inclusive are for 1915 and for a selected group of companies whose records were complete. Thus the figures given in Tables 1 to 3 inclusive do not necessarily check with the totals given for 1915 in Table 5.

TABLE 5. ACCIDENT DATA FOR THE LAKE SUPERIOR REGION, 1911-1915.(a)

	Number employed.	Days active.	Number killed. Total:	per 1,000.	Number injured. Total:	per 1,000.
Michigan:						
1911	31,584	297	134	4.24	7,352	232.78
1912	29,469	304	96	3.26	8,064	273.64
1913	28,174	280	82	2.91	6,521	231.45
1914	27,272	274	99	3.63	6,311	231.40
1915	27,512	289	99	3.60	7.821	284.28
Minnesota:						
1911	16,548	252	76	4.59	6,190	374.07
1912	16,559	287	50	3.02	4,424	267.16
1913	19,546	306	63	3.22	6,360	325.39
1914	17,694	249	43	2.43	3,427	193.68
1915	14,572	274	36	2.47	2,991	205.26
Wisconsin:						
1911	2.844	259	3	1.05	336	118.14
1912	3,834	271	22	5.74	619	161.45
1913	4,330	288	11	2.54	1,236	285.45
1914	2,576	250	11	4.27	487	189.05
1915	2,651	270	8	3.02	605	228.22

aCompiled from annual metal-mine accident reports, U. S. Bureau of Mines.

THE FOUNDING OF THE CALUMET & HECLA MINE 1866-1916.

(NOTE:—The Calumet & Hecla Mining Company published a pamphlet for distribution at the time of celebrating its Semi-Centennial, July 15th, 1916. The facts stated are a valuable record of the early history of the mine.—Secretary.)

The discovery of the Calumet mine was due to an almost inconceivable chance. After piecing together what we now know, we can look back to the events that led up to it with considerable confidence. Several centuries ago, perhaps before a white man ever set foot on American soil, we see a party of Indians digging out copper on Isle Royale. As the winter approached they prepared to migrate to quarters further south; and loading their canoes deep with the spoils of the summer's work they paddled to the mainland toward the southeast. Avoiding the hazardous voyage around Keweenaw Peninsula, they started to carry across it, and were probably overtaken by an early winter storm. At all events, they dug a pit and buried their copper near the place that afterward became Calumet number 1 shaft. By a wonderful accident the spot chosen by the Indians happened to be on the hanging wall of the Calumet conglomerate, close to the lode.

Relieved of their burden, the primitive miners hurried toward a milder climate, but must have met with some disaster on the voyage, as they never returned for the copper. Generations lived and died, little seedlings on the edge of the pit grew slowly into huge trees, and in the stillness of the primeval forest the buried hoard of the Indians guarded in secret the mighty treasure that lay beneath.

The early French explorers found copper on the Ontonagon River in the seventeenth century. But Keweenaw Peninsula was still a virgin forest seventy-three years ago, when it was ceded to the United States by the Chippeway Indians. After the "copper fever" of 1845 and 1846 had subsided, it left the country still a wilderness, with a few small mines hidden here and there in the forest. Such were the conditions in 1858 when E. J. Hulbert was surveying a state road from Copper Harbor to what is now the southwest limit of Houghton County. Roaming about in the woods he stumbled across the pit where the copper from Isle Royale lay buried. This he mistook for an old Indian copper mine. Looking about in the vicinity for further traces of workings, he found, a few hundred feet to the southwest, near what was afterward Hecla number 1 shaft, a large block of conglomerate. This rock, weighing many tons, was infiltrated with copper, and was in fact a bit of the Calumet conglomerate. It was evident from its appearance that it was not a boulder transported there by the ice of former ages, and he was convinced that it had been lifted out of the ground by the frost and lay close to its original position. Satisfied that he had made an important discovery, he went about his business, keeping his secret to himself. A secret that was valuable only because the pit happened to be where it would have been had it indeed been the mine for which he mistook it. Taking the greatest care to keep his actions quiet, Hulbert tried to gain possession of the land on which the pit was situated. With his tiny capital he was unable to do this, but he eventually bought the land to the north. So that it was not until August, 1864, that he found himself in a position to make an opening. Assuming that the boulder was near the lode, and that the pit was an opening into the lode, he drew a line from the boulder through the pit and extended it to the nearest point on his own land. Here, at what is now Calumet number 4 shaft, he sank a shaft, and by the middle of September he was well into the copper-bearing conglomerate. We must therefore consider 1864 as the date of the actual discovery of the lode.

Hulbert's next move was to ship a quantity of the rock to Boston, where he soon went himself. Here he succeeded in interesting Mr. Quincy A. Shaw in his project. Mr. Shaw assisted him in buying from the St. Mary's Mineral Land Company the land to the south of his original purchase. Finally Mr. Shaw and his friends acquired from Hulbert a controlling interest in these lands and further options on this property, which was already known as Calumet.

The final settlement with Hulbert came in the dark days of the history of the mine, when he chose to exchange his Calumet holdings for an interest in the Huron mine. The latter went to pieces and other ventures of Hulbert's were unfortunate. But Mr. Shaw, with unusual generosity, pensioned him handsomely and he passed the last years of his life in Italy, where he lived in comfort to a good old age.

Let us now turn back to the affairs of the mine where we left them to follow Hulbert's fortunes. Shortly after Mr. Shaw and his friends purchased Calumet, the same interest bought the adjoining land to the south, which was to become the Hecla mine. It has not been possible to find any records of the exact dates of these events. Hulbert went back to Michigan to develop the Calumet mine and began to make openings for the company early in 1866. No work was done on Hecla until the fall of that year.

Meanwhile Mr. Shaw's brother-in-law, a young naturalist at Harvard University, was eagerly watching the course of these events. Alexander Agassiz, at that time barely turned thirty, was educated as an engineer, but his inherited love of science had drawn him to his father's museum, where he filled the position of an impecunious assistant. In the summer of 1866 Mr. Agassiz took a vacation from the museum and came up to Michigan to judge for himself of the promise of the mine. On his return he was made treasurer of both companies.

Toward the end of the year it became evident that conditions at the mine were unsatisfactory, that the local management was becoming involved and was not able to make the Calumet mine pay. The openings on the Hecla property had only just been started. The first rock mined was exceedingly rich and the management seemed to have lacked the knowledge of how to mine the rock or to treat it after it was mined. Large open pits on the lode were made which could not be continued for any depth and which would only permit of a very limited output. An attempt was made to smelt the rock; when this failed Mr. Hulbert leased a mill in Hancock, bought one hundred teams of horses and proceeded to haul the rock in wagons about thirteen miles. Such methods began to bear their natural fruit; things went from bad to worse. The hard, tenacious and finely subdivided conglomerate was found to be a very different thing to mine and mill from any rocks hitherto worked in the district. The best experts of the day declared that the lode could not be operated at a profit, and the enterprise threatened to end disastrously for all involved. But there was one man who insisted that it could be made to pay, and that he could do it; this man was young Mr. Agassiz. Finally, when affairs looked very black, the management in Boston decided to give him his chance, and sent him to the Peninsula to relieve Hulbert in the management of both mines.

Mr. Agassiz reached Calumet early in March, 1867. The camp consisted of a collection of shanties, and the so-called hotel in which he took up his headquarters was little more than a cabin on the edge of the forest. Mrs. Agassiz, who joined him later, wore a pistol when she went out walking with her little boy. When she left, the baby's perambulator was passed on to the younger brother of the present general manager. Her youngest son, now at the head of the company, was not then in existence.

In order to fully appreciate what Mr. Agassiz succeeded in accomplishing during his residence at Calumet, it is well to remember that the Keweenaw Peninsula was in those days as inaccessible in summer as Alaska is today; and after the close of navigation the region was practically cut off from the rest of the world. For Green Bay was then the terminus of the railroad, from where it was a trying sleigh ride of many days to Calumet.

Mr. Agassiz found on his arrival that practically nothing had been done to develop either mine along legitimate lines. What little had been done had been done wrong. In one of his earliest letters to Mr. Shaw he complained that there seemed to be no supplies of any kind on hand except hay and oats.

Under such trying circumstances, with many hands against him, and looked on with distrust as an inexperienced outsider. all things had to be started afresh, and the mistakes of the former mismanagement corrected. The old letter books, with their faded copies of the correspondence between Mr. Agassiz and Mr. Shaw, reveal in some measure the gallant team-play of the two men, as they strove against what seemed hopeless odds to get the mines on their feet. Mr. Agassiz, with insufficient means, was trying to establish order out of chaos at Calumet; Mr. Shaw, in Boston, loaded with debts, and saddled with the collapsing Huron mine, was struggling to obtain funds from a community that had lost confidence in the enterprise. On more than one occasion they appear to have been on the point of being forced to give up the fight, and sell out for what they could get. As Mr. Agassiz wrote years afterward: "If Quin had ever known when he was beaten we should never have pulled the thing off."

All day long Mr. Agassiz rushed from one place to another. "The thing I drive and look after is the only thing that goes," he writes Mr. Shaw, "and just as fast as I pass from one thing to another, just so fast do things move. I ought to have had three good men instead of being compelled to do all I have to do myself. There is not a thing, down to seeing that the cars* get unloaded when they come here with materials, which I don't have to look after myself, and some days I am in utter despair." After such a strenuous day, he would sit at his desk far into the night writing business letters, straightening out the accounts, and planning how best to make every cent tell. Mr. Shaw meanwhile was financing the mines as best he could in Boston, and receiving substantial aid from Mr. John Simpkins, for many years the selling agent of the company.

Mr. Agassiz lived nearly two years at Calumet. During that time he succeeded in properly reopening and equipping two complete mines in the wilderness, besides building and fitting out a mill at Calumet, fed by the little stream that flowed through the village, and a similar mill at Torch Lake for the Hecla mine. Each mill was provided with two ball heads and the necessary foundations and provisions for installing two more. In addition he constructed four and three-quarters miles of railroad through the forest to Torch Lake, besides the various connections at the mines, and dredged a communication with the navigation of the Great Lakes.

By the early summer of 1868 daylight began to appear and the endless difficulties commenced to straighten out. The last serious setback was the cutting of the Calumet dam by some men in the employ of Agassiz's enemies. But by this time the intruder from the East had won the confidence and support of the community and a willing crew was rushed to the repair of the dam, and all was soon running smoothly again. By the end of the summer two prosperous little mines were producing between them about 325 tons of ingot a month. And before the close of navigation the conditions were such

*This refers to the cars of the C. & H. railroad to Torch Lake. There were, of course, no outside railroad connections at that time.

that Mr. Agassiz felt satisfied to leave the mines in charge of Captain George Hardie.

It would be entirely outside the province of this little pamphlet to attempt any description of the gradual growth of the mines from such comparatively small beginnings to the tremendous industry that it has today become. An industry which in some months handles the vast output of 11,000 tons of rock a day.

Hecla paid its first dividend on December 15, 1869, and Calumet followed suit on August 5, 1870. In May, 1871, the Calumet mine, the Hecla mine and the Portland and the Scott Mining Companies were consolidated into the Calumet & Hecla Mining Company with Mr. Shaw as its first president. In August of the same year he retired to the board of directors and Mr. Agassiz was elected president, a position which he held until his death in 1910. Mr. Shaw and Mr. Agassiz directed the policy of the mine until a few years before the former's death in 1908. As Mr. Shaw had no training as an engineer, the development of the mine itself fell to Mr. Agassiz, who always visited the mine twice a year, assured himself of the actual conditions there and the proper execution of all the work planned.

On Captain Hardie's retirement as general manager he was succeeded for a short period by Mr. T. W. Buzzo, who gave place to Mr. R. J. Wood, elected in the fall of 1871. The next man to fill this position, Mr. James N. Wright, served from his appointment in May, 1873, until his resignation on January 1, 1892. Mr. Wright was succeeded by Mr. S. B. Whiting, who resigned in 1901. Owing to his ill health, work at the mine was at times directed by his assistant in charge. This position was filled by Mr. J. P. Channing in 1893-94, and by Mr. S. D. Warriner from 1897 to 1901. Mr. James MacNaughton, the present general manager, who has grown up with the district, was appointed in 1901; and the management of the Boston office has fallen on the shoulders of a younger generation.

It is impossible to give here an adequate acknowledgment of the work of the many men who have helped to make the Calumet & Hecla what it is today. And with full recognition of the able and faithful services of others, it must be recognized that Alexander Agassiz's was the guiding hand that directed the evolution of the mine.

The growth of the Calumet mine was founded on his policy of looking ahead to see what the conditions would be years later, to make ready for them far in advance, and to keep the mine opened up a long period ahead of the work. For instance, the "Superior," an engine installed in 1883, was designed to hoist six skips, each with a capacity of four tons of rock, from a depth of four thousand feet, and also to run four Rand compressors. As the mine was then hoisting two and a half ton skips from an average depth of about fifteen hundred feet, the engine was greatly in excess of the needs of the day, and was considered by many people a white elephant. In 1911 it was hoisting five ton skips from a depth of six thousand feet!

Mr. Agassiz never hesitated to spend money freely for a future return, and to build not only for the coming years, but for the next generation. A less enlightened method would never have enabled the company to handle so economically today such vast quantities of low grade rock as it is now hoisting from the depths of the mine.

Such a policy naturally incurred great expense, and there has been no little criticism in the past of the extravagant management of the mine, by those who were unable to see the benefits ahead. It is worth emphasizing that many of these complaints came from the very men who had previously declared that the Calumet conglomerate could not be worked at a profit.

What Mr. Agassiz accomplished was in a great measure due to his ability to handle men, to make them like and respect him. And if they worked faithfully for him, it was with the knowledge that he was working for them. For side by side with his development of the mine he devoted his best efforts to promote the comfort and well being of its employes. He strove that the hospitals, doctors and schools should be the best of their kind. He saw that there were comfortable houses for all, he established an aid fund, and helped build the churches. It is due to his efforts that the community is looked on as a model today wherever intelligent men are striving by sane methods to improve the conditions of American citizens. Some years ago the governor of Michigan, in speaking of the labor conditions of the state, said that Alexander Agassiz had done more than all others for humane and reasonable conditions of life among its people.

One of the main objects of the celebration of the fiftieth anniversary of the opening of the mine is to endeavor in some measure to recognize the great part that the men have played in its successful development. The chief event is the presentation of commemoration medals to those who by long and faithful service have built the Calumet & Hecla.

A few decades ago this country was an unknown forest. At the fiftieth birthday of Calumet we see a prosperous community of some fifty thousand souls dependent for their well-being on a wonderfully successful mine. Such a change was not wrought by a party of sybarites who drifted into this district in a parlor car. It was hewn from the wilderness by determined men, who, fighting through darkness and gloom, forced their way into the light, brought peace and plenty to thousands of working homes, and created one of the most famous mines known in the history of industry.

ELECTRICAL POWER IN MINING ON THE MENOMINEE RANGE.

BY CHARLES HARGER, IRON MOUNTAIN, MICH.*

There are three large hydro-electric power developments on the Menominee Range supplying power almost exclusively to the iron mines. Two of these systems are owned and operated by individual mining companies. The third is that of the Peninsular Power Company, which sells its output to mines throughout the range. All three plants are on the Menominee river, which is on the state line between Upper Michigan and Wisconsin.

THE PENN IRON MINING COMPANY'S PLANT.

The oldest of the three systems is that at Sturgeon Falls, operated by the Penn Iron Mining Company, the total output of which the Company uses in its own mines in and about Vulcan. The equipment at the power station consists of two water-wheel-driven generators with a total capacity of 3,500 k.w., and a steam turbo-generator of 1,500 k.w., which is used in case of low water, ice or other difficulties at the dam, or transmission line trouble. The Company's entire mining equipment is electrically driven. When the transfer was made from steam to electric power, new apparatus was purchased or else the original machines were converted to electric drive. Among the latter are several hoists and an air compressor.

The hydraulic plant was very fully described in a paper presented before the Lake Superior Mining Institute by Messrs. T. W. Orbison and F. H. Armstrong in 1908. The efficiency of this electrically driven mining machinery, with complete data of tests, together with a full description of the converting of the old steam equipment to electric drive, was presented in a paper by Messrs. Wm. Kelly and F. H. Armstrong, at the New York meeting of the American Institute of Mining Engineers, in February, 1914.

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*Superintendent of Peninsular Power Company.

ter-wheel-driven units of 1,100-k.w. capacity each. When the success of electrical operation became assured, and the mining companies became aware of the economy, flexibility, simplicity and convenience of this kind of power, the business of the power company increased rapidly, and, during the winter of 1915-16, two additional units of 1,250 k.w. each were installed at Twin Falls, making a total of 5,800 k.w. available at this station. From the small output of 25,000 k.w.-hr. in the month of May, 1912, the load has increased to 1,600,000 k.w.-hr. in December, 1916, the peak load increasing from 200 k.w. at the beginning to 4,500 at present.

With the exception of 500 k.w. used by the cities of Iron Mountain, Iron River, and Florence, and 50 k.w. used by the Iron River Street Railway, all power produced is taken by the mining industry. Power is now being delivered in various quantities to thirteen of the large mining interests of Iron River, Crystal Falls, Alpha, and Florence, serving twenty-one mines. Lines are now under construction to two other mines in the Iron Mountain district.

As the present load is more than the power available from the water power in times of dry weather, the company has added correspondingly to its steam plant and now has steam turbines installed at Iron River with a total of 3,500 k.w. capacity. During the past six months, negotiations for the purchase and the preliminary steps in the construction of a second water power have been rapidly going forward, in anticipation of more business as new mines are opened or the present steam-operated mines change to electrical equipment. Eighteen months from now, double the present load, it is hoped, may safely be carried by water power, except in extremely dry weather, when the steam plant, also to be enlarged, will be put in operation.

The present high-tension transmission consists of forty miles of 66,000-volt lines connecting the power plant at Twin Falls with the mining districts of Iron River, Florence, and Alpha. The line is built in duplicate and supported on steel towers set in concrete bases. Current is delivered to the mines from the Company's three high-tension substations at 6,600 volts on duplicate lines also carried on steel towers. At each mine a secondary substation is installed and maintained by the power company, the mining company furnishing the building. The equipment at these low-tension stations consists of three transformers which step down the voltage of 2,200, lightning arresters, an automatic oil circuit-breaker with the necessary relays, and meters mounted on a panel. Power is metered and sold to the customer at 2,200 volts. With but few exceptions all motors of 25 h.p. and over are of this voltage.

A detailed description of the complete development was given by Mr. C. V. Seastone, consulting engineer and vice president of the Peninsular Power Company, in a paper presented at the New York meeting of the American Institute of Mining Engineers, in February, 1915.

Of the twenty-one mines supplied by this company, seven

FIGURE 2. TURBO-GENERATOR ROOM AT IRON RIVER.

are completely electric, steam being used only for heating the buildings. The equipment varies from 400 h.p. at the two mines of the Florence Mining Company at Florence, to 1,400 h.p. at the Corrigan-McKinney Company's new Odgers mine at Crystal Falls. All compressors, pumps, hoists, crushers, blowers, and haulage and shop machinery are driven by electric motors. With the exception of the crushers and the compressors at the mines operated by the Florence Company, all machines are direct-connected to the motors. In the Florence equipment, the air compressors are belted. The remaining fourteen mines, which are only partially electric, were steam equipped before the power company came into the field. The equipment which has been changed from steam to electric in most cases consists of electric pumps and motor-generator sets for underground haulage, in some cases of pumps or generator sets only, and in one instance, that of the Bristol mine at Crystal Falls, of an air compressor only. The rest of the equipment is steam driven at each mine.

Power Contracts.

The power company's contracts with all mining companies are identical, varying only in the stipulation of the minimum charge, this being proportional to the amount of power guaranteed and upon the rated connected load. The rate per kilowatt hour is based upon the load factor of the load, ranging from 1.49c at a 20% load factor to 0.9c at 90% and above. The one-cent rate covers a range of from 50 to 80% load factor. The load factor is determined by the use of a maximum-demand meter registering 3-minute peaks and so connected that the hoisting load is not registered by the demand meter. With this arrangement a mine electrically equipped throughout with a small amount of care and attention in operation, may easily keep within the one-cent rate. Where only the pumping is electric, the load being fairly constant and continuous, the low rate of 9 mills may be attained.

Operation.

Like all new problems, which present themselves in mining, electric operation requires considerable study and attention on the part of the superintendent in charge. When power is purchased and paid for on a load factor basis, the load must be carefully regulated to get the desired high load factor. When only pumping is to be considered, careful plans should be made in the design of the installation. As the output of an electric pump can be varied only by by-passing the water or by stopping and starting the pumps, either energy is wasted in the first case, or the load factor is impaired in the latter, if the pump is of too great capacity.

Therefore, rather than install a pump of too large capacity, it is more economical to put in one that will barely take care of the water under normal conditions and have a smaller pump to use intermittently when the sump becomes full. When an auxiliary pump is used in this way, or in fact whenever any auxiliary machine is used, care should be taken to open the switch controlling the running unit before starting the auxiliary; this holds the peak down and keeps up the load factor.

Where a large sump is available and the amount of water is not too great, the pumping may be done at off-peak periods, when the compressor is idle or at light load between shifts.

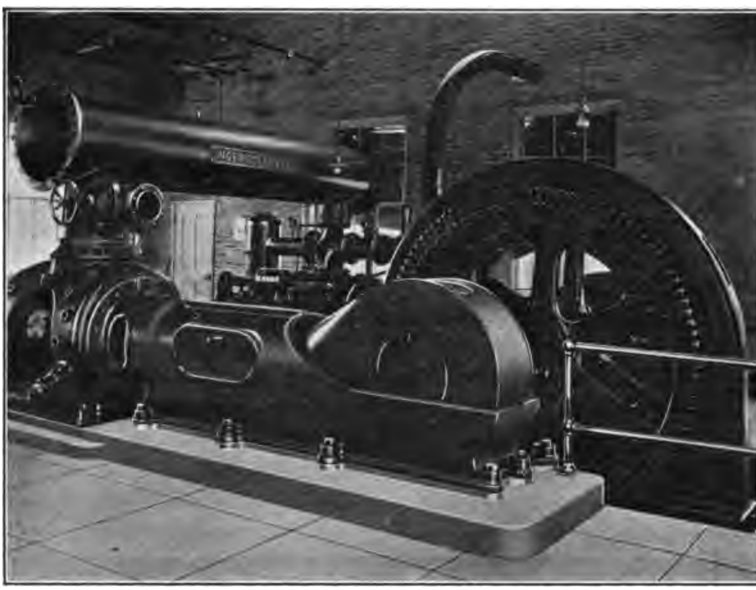


FIGURE 3. AIR COMPRESSOR AT DAVIDSON MINE.

Again care should be taken to delay starting the pump until the load is off the compressor. This method is used very successfully by Superintendent Ericson at the Davidson mine at Iron River.

In the case of a new mine where water conditions are not yet fully developed, the following plan is successfuly used by the Verona Company: A reciprocating pump is installed of larger capacity than the present water demands. The cylinders are bushed and smaller pistons inserted. As the mine develops and the water increases, the cylinders are made larger by changing the size of the bushings and plungers or perhaps removing the bushings entirely and using the pump at full capacity. At present there is being developed a pump so designed that the length of stroke may be quickly and easily changed to meet the above conditions. Also, without doubt, there will shortly appear on the market a motor-driven pump with a gear-changing arrangement similar to that of an automobile, by which the speed of the pump may be changed without changing the motor speed. When this is done, the electric pump will be as flexible as the old steam pump.

As all compressors connected to this system are driven by synchronous motors, almost ideal voltage regulation is possible. All compressors may be blocked at one-fourth, one-half, three-fourths, or full load. This arrangement is also taken advantage of by the wide-awake superintendent or engineer in holding his load as nearly continuous as possible.

In the matter of load regulation, a graphic or curve-drawing wattmeter should be a part of the switchboard equipment in the power house of each mine. Thus the man in charge will have a complete record of his load each day, which will very materially help him in arranging his operation so as to keep down the peaks and level the curve. The meter is also a check on the power company's demand meter. The meter should have a chart travel of four inches to the hour. Equally desirable is a watt-hour meter connected to each circuit feeding the several machines so that the cost of operating each unit can be computed and distributed. The total of the several meters is a fair check upon the main watt-hour meter of the furnishing company.

Construction and Installation.

In any construction work the thought of curtailing first cost generally predominates. In the electrification of mining equipment, this idea should not be carried to an extreme which does not leave the installation thorough in construction, secured against interruption, ample in capacity, and, above all, safe, simple, and convenient in operation. The engine house, containing the hoists, compressor, motor-generator, and switchboard, should be given thoughtful consideration in the arrangement. The floor space taken by electrically driven machinery is less than 50% of that required for steam equipment of the same capacity; but the equipment should not be too crowded. Among the ideal power houses served by this company are those at the Bengal mine of the Verona Company, the Homer mine of the Buffalo Steel Company, the Davidson mine of the Davidson Ore Mining Company, the Berkshire mine of the Brule Mining Company, and the Odgers mine of the Corrigan-McKinney Company.

The construction underground is of prime importance. As the operation of a mine is primarily dependent upon the pumping equipment, this part of the installation should be carefully planned. A large room for the pumps should be provided, and, as moisture is the worst enemy of an electric

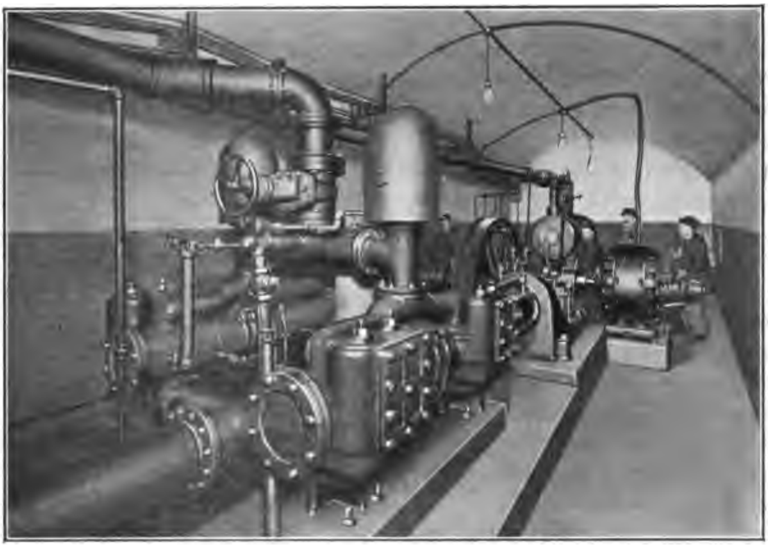


FIGURE 4. ELECTRIC PUMPS AT BENGAL MINE.

motor, the room should be made as dry as possible. With but few exceptions, all underground pump motors on this range are of 2,200 volts. Operators were at first apprehensive about using current of this pressure underground, but it is now almost the universal practice. With the modern method of conducting the current into the mines, the danger of injury from contact with the conductor is removed. The most approved method is to use a 3-wire, varnished-cambric or rubber-insulated, lead-covered, jute-wrapped, steel-armored cable. A special form of hanger is made which securely clamps the steel armor without injury, so that when the cable is in the shaft all the weight is carried by the armor. If this cable is taken from the bus at the engine room down the shaft and into the pump station, with no taps or other openings of the insulation in its entire length and thoroughly protected at the switch, there is no danger of rupturing the cable, all danger to workmen is eliminated, and continuity of service is assured. The most perfect of all pumping installations is found in the great Chapin mine of the Oliver Iron Mining company. Other first-class stations are at those before mentioned at the Bengal and Davidson mines, and at the Tully mine of the Corrigan-McKinney Company.

It is not the object of this paper to discuss in any detail the type of apparatus to be used. The topic of the efficiency and maintenance of reciprocating and centrifugal pumps has been discussed pro and con before this and other mining associations and the writer will merely state a few observations gathered from his contact with the engineers and mining men connected with this power system. In the matter of cost of power in operation, the plunger pump surely has the preference on the Menominee Range. However, where the items of maintenance, first cost, space required, and vibration of the discharge column are considered, the centrifugal pump is as usual in the lead. Also, as is generally the case, with the higher heads and larger flows of water the centrifugal pump is more efficient and consequently cheaper in operation than at lower heads and smaller water volumes. This matter of type of pump to be used is, therefore, one to be decided by individual cases. Mr. C. E. Lawrence states that, at the Balkan mine at Alpha, where the surface was stripped to the depth of ninety feet and much surface water had to be handled. it would have been next to impossible to have completed this work on time had it not been for electrically driven centrifugal pumps. Several of these pumps, of small capacity, direct connected to the motors on temporary timber foundations, were used at various points about the pit. They were moved about to suit conditions and kept out of the way of the drag shovels, sometimes being moved several times each week. When the project was started an attempt was made to handle the water with steam pumps, but after a few feet had been excavated, the increasing water, the caving sand and the frequent moving of pump sites, so hindered the digging contractors that the steam pump was discarded and the electric pumps put in. When this was done, to use Mr. Lawrence's words, "there was nothing to it."

COST OF OPERATION.

Below is given data on the cost of current in electric pumping at several mines of the Range. The rate in all cases was one cent per kilowatt hour, as computed from information supplied by the consumer. The figures are taken from actual operations covering periods from 24 hours to 30 days.

Mine.	Company Operating.	Head in Ft.	Gal. per Min.	Type of pump	Cost per Million Ft. Gal.
Bates	Florence Mining Co.	470	840	Centrifugal....	5.12 ct.
Fogarty	Verona Mining Co..	368	315	Reciprocating.	4.52 ct.
Bengal	Verona Mining Co..	280	168	Reciprocating.	4.62 ct.
Tully	Corrigan-McKinney.	508	1,140	Reciprocating.	4.19 ct.
Davidson	Davidson Mining Co	490	168	Reciprocating.	4.43 ct.

The Davidson mine has a 580-gallon pump running about seven hours each day at off-peak times.

The writer has also taken the liberty to give the computed cost at one cent per kilowatt hour in four months' operation of the centrifugal pumps on the twelfth level at the Chapin mine. The data, from which this is figured, is taken from a communication from Mr. S. S. Rumsey to the Secretary of the American Institute of Mining Engineers published on page 807 of the 1914 proceedings. The total number of gallons pumped during the four months was 443,886,000 measured by a Venturi meter, an average of 2,830 gallons per minute. The lift was 966 feet, the amount of current used 1,938,120 kilowatt hours, the cost per million foot-gallons 4.55 cents.

It is interesting to compare these figures with the results of an efficiency test made some time ago at one of the most elaborate and modern steam pumping plants on the Range. All the pumps at this plant are triple-expansion, handling 3,500 gallons of water per minute under a lift of 335 feet. Modern water-tube boilers are used and the steam lines are thoroughly insulated. The test was of eight hours duration with the complete equipment in prime condition. The coal cost alone during the test, computed at $4 per ton, amounts approximately to the average total cost of power for the electric pumps above. In determining the total cost of the above pumping, certain costs will have to be added which do not enter into the cost of electric pumping, such as boiler-room labor, the maintenance of the boiler equipment, the steam line and the steam end of the pumps, and the large overhead charge which goes with an expensive layout of this kind.

There is no longer any question as to the economy of electric pumping. Ninety per cent. of the mining men of the Menominee Range will so testify. At mines where individual meters are not connected with each machine the saving is plainly noticeable the first day of the operation of an electrically driven pump by the decreased amount of coal used compared with increase of current used as shown by the power company's meter. In a mine with the ordinary amount of water, electric pumps can be installed, the steam pumps being left in the mine for auxiliary or relay use, and the cost of the new equipment will be saved within a few months.

The electric air compressor, while not as economical as the electric pump, can be operated at a saving when used in conjunction with the rest of the equipment. The load of an air compressor alone is generally of low load factor, due to its operation but part of the time and the constant friction load when running light. However, when the whole equipment is electric, the load factor is maintained at 50% or above.

Experience shows that hoisting by electric motors is ideal. There are seven electric hoists in the Range which are hoisting ore at an average of 1.4c per thousand foot-tons. A brakeman experienced on steam hoist soon adapts himself to the new power without trouble. It is generally known that higher hoisting speeds may be safely attained by electric power than by an engine-driven hoist. This is due to the safety appliances such as the magnetic brakes and governor-controlled automatic circuit-breakers which are used on modern electric hoists. These safety features are recognized by the German government in their statutes regulating speed limits of different types of hoisting machinery.

The economy, practicability, convenience and adaptability of electricity in mines has been proved beyond the shadow of a doubt by the mining industry of Northern Michigan. With a motor, which is well installed and well taken care of, there is practically no depreciation and but little maintenance. The greater number of the new mines in the future will be of electric drive. With the prevailing price of coal, mining companies in regions as far removed from the coal supply as the Menominee Range is, may consider themselves fortunate if they lie within delivery distance of hydro-electric power, with fair assurance of continuous service at a reasonable rate.

CONCLUSION.

The writer, although not a mining man, wishes in conclusion to give a brief description of what he considers the ideal appearance of a mine. Upon approaching this mine one admires the grounds. Instead of the usual soot and smoke-covered landscape, a green lawn is noticed in which neatly plotted flower beds are conspicuous. There is no smoke-stack belching forth high-priced coal, no coal dock, no ash heap, and no exhaust steam to be seen. The inactivity about the place leads one at first to believe the mine closed down or abandoned, but closer observation reveals sheaves at the top of the head-frame revolving seemingly without power. Also tram cars are running out on the stock pile filled with good red dirt, with no visible means of propulsion. Upon entering the engine room the absence of noise is noticeable. There is no clanking of valve gear, no hissing of steam and no unsightly oil pans about the compressor. Instead, there is the hum of a smooth-running motor between the high and low-pressure cylinders constantly delivering the air. The brakeman, who is the only man in the room, touches the controller and the drum of the hoist begins to roll, noiselessly and with no other moving parts to be seen. Everything about the room is spick-and-span, brightly painted and polished.

Underground, the conditions are no less pleasing. As we descend in the cage, lowered by the electric hoist, bright lights greet us at every level. On the lower level, a train of cars loaded with ore and drawn by an electric mule is approaching the skip pit. As we enter the pump room we are astonished at its appearance—instead of the usual dark, damp, steam-reeked hole in the rock, we enter a room fit for a banquet. The air is fresh and without the usual intense heat. Bright lights are shining, walls are brightly painted and decorated with pictures. Here are motor-driven pumps constantly discharging to surface the much dreaded water and thus insuring the profitable mining of the property.

Anyone acquainted with the Iron River district will recognize in the above description the Bengal mine, under the general supervision of Chas. E. Lawrence, President of the Lake Superior Mining Institute.

REMINISCENCES OF THE DEVELOPMENT OF THE LAKE SUPERIOR IRON DISTRICTS.

BY JOHN M. LONGYEAR, MARQUETTE, MICH.*

I have been requested to write some reminiscences of the development of the Lake Superior Iron Districts, with which I have been more or less connected, and am writing this paper in compliance with the request of the committee.

My first acquaintance with the Lake Superior region was in 1873, when, on the first day of June, I arrived in Marquette on the steamer "Rocket", having passed through forty miles of ice between Grand Island and Marquette. At that time the Lake Superior Iron district was Marquette County, there being only one small mine just over the line in Houghton County (now Baraga County), the Spurr mine.

In the year 1873, the total shipments of iron ore were about 1,250,000 tons, and there were "calamity-howlers" who deplored the wasteful extravagance of shipping so much iron ore in one year, and ruining the market. The schooner Pelican had taken, during that summer, a record cargo of 1,250 tons, and there were those who protested against the attempt to use such large vessels in the ore trade, claiming that large vessels like the Pelican could never be made to pay on the lakes. In view of the recent outputs of the iron ore districts of Lake Superior and of the size of the ordinary cargoes now carried by vessels on the Great Lakes, these figures seem absurd, but they were taken very seriously by many well-posted people in 1873.

Marquette was a thriving little city of about 3,000 population, and was the only port on Lake Superior from which iron ore was shipped. An ore dock had been built the previous year at L'Anse, but was very little used and was demolished a few years later.

In 1873, the upper peninsula of Michigan was an undeveloped wilderness and nearly all covered with a heavy growth

(*General Land Agent.)

of standing timber. There were a few towns scattered around the edges of the lake shore, and the manufacturing of lumber had been inaugurated at several points. For some years railways had been in operation between Marquette and the mines at Ishpeming, Negaunee and Champion. The Republic mine had also been connected by rail a year or two before. The opening up of the Republic mine had created much dissatisfaction in some circles; their argument being that it was useless to build a railroad to a remote mine like the Republic, when there was all the ore in the Ishpeming district the country could use for generations.

In 1873, the only railroads in the upper peninsula were the lines from Marquette to L'Anse, from Negaunee to Escanaba and Menominee, and the narrow-gauge road between Hancock and Calumet. Besides these there were a few short spurs to mines.

There were very few wagon-roads, and these generally of the roughest character, commonly known as "tote-roads." These generally extended only a few miles back from the lake shore. Agriculture was considered to be an impossibility in the Lake Superior region, and it was some years before we begun to understand that the region had a great asset in its agricultural land, which the many fine farms developed during the last few years have fully demonstrated.

In visiting the undeveloped regions, the only means of access was by walking, even pack-ponies were impossible, so it was necessary to carry everything we used, except fuel, air and water by packmen. The explorers, or "bush-whackers," as they were often called, were sometimes obliged to go fifty or more miles in the wilderness from any settlement, and it was impossible to carry more than about two weeks supplies in one pack-load, so short and hurried trips were the rule. The most of the exploring was done for pine timber only, the other varieties of timber at that time not being considered of any value. A little pioneer work was done in the way of looking up indications of iron ore, but without much result.

During the summer of 1873, on the Lac Vieux Desert trail, I met one party of three men who were going into the wilderness to explore for gold. They had heard that gold had been found on the head-waters of the Ontonagon river, and they had set out to find it. The head-waters of the Ontonagon river probably drained a thousand square miles of territory, practically every mile of which was covered with a dense forest. Evidently their undertaking was one of large dimensions. I found that these men were entirely ignorant of any knowledge of woodcraft. Their supplies consisted of about ten pounds of soda-crackers for each man. Only one of the party had ever seen a gold mine, and I learned that he had chopped wood for three months for a gold-mining company in California. That was the extent of his mining or geological knowledge. Our party, and these explorers also afterwards considered that we were agents of Providence in rescuing them by piloting them back to the settlements, as they doubtless never would have found their way out of the woods had they not chanced to meet us. They had provided themselves with light moccasins of the type usually sold to tourists, which they wore like a shoe over a single light sock, and after a few days in the woods their feet were swollen to unbelievable proportions, which swelling continued until after they left us at Houghton, where they were unable to buy foot-gear large enough to accommodate their feet They finally bought the largest sized carpet slippers, split them on top and tied them on with strings, managing in this way to provide foot-gear in which they could walk.

THE MENOMINEE IRON RANGE.

My first visit to the Menominee Iron Range was in 1874. while I was engaged in examining the scattered tracts of state land, known as the State Mineral Reserve; certain state lands having been withheld from market for possible mineral values. On the request of Governor Bagley, Mr. A. B. Wood of Owasso, Michigan, and I were examining these lands for the state. We left Republic in a flat-bottomed skiff, and proceeded down the Michigamme River, making numerous portages and side trips to visit the reserve lands lying west of the river. At the Grand Portage, about three miles above the mouth of the Michigamme River, we lost our boat and the rest of the journey was made on foot. We camped one night at the Twin Falls on the Menominee River where the hydro-electric station of the Peninsular Power company now is. A tote-road had been brushed out to this point, and the tracks of the one wagon which had penetrated that far into the wilderness, a few weeks before, were visible.

We saw some log camps at the explorations which afterwards became the Vulcan and Emmett mines. At the latter, where the village of Waucedah now is, we found a caretaker who put in his abundant leisure raising potatoes in the clearing around the camp. We were much interested in this experiment, as agriculture was a new thing in the Upper Peninsula. We were told that there was not a farm within sixty miles of this camp, but the caretaker had a bushel of dead potato-bugs he had harvested from his small field.

From Waucedah to the railroad at "42," now Powers, we followed a winter road through a dense cedar-swamp, the surface of which was from six inches to three feet under water. It was hard walking for we could not see how deep the water was, but it was the best going there was. The route of this road was near the present line of the Chicago and Northwestern and the excellent highway. The swamp is mostly cleared away and many excellent farms occupy the greater portion of the former extensive cedar-swamp.

Iron ore had been reported in various places on the Menominee Iron Range when the U. S. linear surveys were made about 1850. Some exploring had been done in 1868 to 1870, but nothing of importance had been found. The early explorers had based their work and opinions of the district on their knowledge of the Marquette district, and it was several years before they realized that the formations were quite different.

The Menominee Mining company composed of Milwaukee and Menominee people were the first to take any practical steps in the way of developing the district. Their first request for a branch track from the Chicago and Northwestern Railway was refused. The railway people had consulted one of the prominent explorers who had done some work on the range, and he had expressed the opinion that there was not enough ore in that district to justify building a road. The Menominee Mining company insisted on having the railway, and the officers of the mining company finally personally guaranteed the Chicago and Northwestern Railway company against loss should they construct a branch about twenty-four miles in length from Powers to the present station of Quinnesec. Powers was then known as "42." The branch road was finally built in 1876-77, and my first acquaintance with the development of the region was in 1877, during which time I made several visits to Quinnesec, riding on a construction train from Powers. There were no regular passenger trains until the following year. At that time the Menominee Mining company had camps at Waucedah, Vulcan and Quinnesec, and for several years Quinnesec was the terminus of the road, and the headquarters for the extensive lumber operations in that district, and all the explorers and miners engaged in the development of the iron ore deposits.

In 1879, I published the first private map of the Menominee Range, which then embraced Towns 39 and 40 N., Ranges 28, 29, 30 and 31 West.

The first option for an iron ore lease issued by me was given to Hon. S. S. Curry, on the W½ of the NE¼ of section 9, T. 39 N., R. 29 W. This property had been explored by Dr. Credner, a German geologist, working for the Canal Company in 1868. He had dug a trench from the bottom of the hill, exposing the formation continuously for several hundred feet. Then, to avoid the difficulties of digging among the roots of some very large trees, he had offset to the west about one-hundred feet, and continued his trench across the formation, apparently exposing the entire ore-bearing belt. Mr. Curry, looking this work over one day, wondered why the pine trees which occasioned the offset were so much larger than the other timber in their vicinity, and as a preliminary to answering this question to his own satisfaction, he applied for and obtained the option above mentioned.

On returning to the property he set his men to work digging a pit between the trees, and within twenty minutes the men were shoveling out a high-grade iron ore, and within two weeks iron ore was being shipped from the property, which was named the Curry Mine. The ore being softer than the adjoining hard formations furnished more nourishment for the roots of the trees, which accounted for the greater size of the trees, and which had caused Dr. Credner to miss the good ore altogether.

From 1878 and through the early eighties, a very large amount of exploring was done in the Menominee district. In 1880 the railroad was extended to Florence, Wisconsin, where two mines, the Florence and the Commonwealth had been developed, and the famous Chapin mine at Iron Mountain had been opened. The Norway and Cyclops mines were opened by the Menominee Mining company in 1878 and 1879. The

Cyclops was not a large deposit but yielded a very high-grade ore. The Norway was a large deposit of high-grade ore, but not so high as the Cyclops.

A townsite was needed near these mines, so I platted a piece of high rolling ground south of the railroad, forming a very desirable site, but it took me so long to get my plat executed by the company owning the land, that J. B. Wymer, who owned the surface rights in the swamp just south of the Norway mine was able to record his plat of that property, and start his town before I was ready. A corduroy road was laid through the wet swamp, on the main street, and buildings were erected with foundations of cob-house piers built of spruce poles, on which the sills of the buildings were laid. In the summer the water beside the corduroy, and under the buildings was covered with a green scum, which did not seem to trouble the inhabitants, for the town grew rapidly. The first sale of lots in my plat was to a stranger who paid cash for two lots near the center of the plat, and not long afterwards I discovered that a large building had been erected on this property to house one of the worst kind of mining-camp dives. It required a great deal of time and considerable money to rid the property of this nuisance, which effectually killed it as a townsite.

Subsequently the great Aragon mine was found underlying the swamp and the townsite, so it became necessary to move the town south to the site I had platted and land adjoining on the west, the present location of the thriving City of Norway.

I think it is fortunate for the dwellers in the City of Norway that ore was discovered under the former site, thus driving them into a much better place.

In 1881, I had a magnetic survey made showing the dip-needle attractions over the iron formation from Waucedah to the Menominee River, northwest of Iron Mountain, distance of about eighteen miles. This survey was of considerable service in laying out exploring work along the Range. The line of maximum attractions being south of the ore deposits made an excellent guide in planning exploring work.

My first visit to the Crystal Falls Iron District was in 1874, while examining the State Mineral Reserve lands, but at that time I saw no indications of iron ore. I was again in the same township (43-32) in 1877, when I noted indications of iron-formation southeast of the present city of Crystal Falls. In 1877, extensive forest fires had damaged much pine timber in this region, and I visited it for the purpose of learning the extent of the damage to our property. At this time there were no roads or trails in that district. In crossing the pine plains southeast of Crystal Falls near the Brule River, I met a party of lumbermen making a tote-road north from the Brule River to Paint River in section 28, T. 43 N., R. 32 W., where the lumbermen were preparing to cut and save their damaged timber. This was the first road into the Crystal Falls district.

In 1880 or '81, I visited the property which afterwards became the Mastodon Iron mine. Here, in a small swamp on swale, I saw an outcrop of clean, hard hematite ore, rising four or five feet above the surface of the ground, and perhaps seventy-five or one hundred feet in length. Standing on the outcrop of ore was a large dead cedar tree, which had been blazed years before, and on the blaze was written in red chalk "Bojoo Nitche" (Chippewa for "Good morning my friend").

The first good ore found in the Crystal Falls district was on the property now known as the Great Western mine. Another exploration was a "chimney" of hard hematite found beside the falls, which gave name to the city and district. This chimney was supposed to lead to a large deposit, but it was entirely mined out and found to be only about eighty feet deep, and not more than twenty feet across in its largest dimensions.

I made three trips in boats down the Michigamme river, one in 1874, one in 1879; the last in 1880. On the last trip we portaged the boats from the Michigamme river by Lake Mary to the Paint river, below the Crystal Falls. This time I met Hon. S. S. Curry in an exploring camp near the falls, and at that time saw the pit revealing this chimney of ore.

The Gogebic Iron Range.

United States linear surveys made about 1850 mentioned outcroppings of iron ore on what is now known as the Gogebic Iron Range, in Section 7, T. 47 N., R. 45 W. The next official reference to iron ore in that region was made in the State Geological report in 1873, in which Brooks and Pumpelly devoted two pages in describing a reconnaissance in that region a year or two previous. Mr. Brooks told me in the spring of 1882, that at the time of the reconnaissance, he and Pumpelly did not think much of the district. At the time of their visit no Lake Superior ore was considered of value, except the speculars and magnetites. The furnace operators were beginning to use the hematites and limonites in 1873, when I made my first acquaintance in the Lake Superior district, but they were used with much hesitation even by the more courageous furnace operators, and the soft ores were then derisively designated "Lake Superior mud." In 1877, acting for the Lake Superior Ship Canal Railway & Iron Company, I employed Mr. Frank H. Brotherton, of Escanaba, to examine the timber and surface mineral indications in Township 47 N., Ranges 44 to 47, inclusive. I subsequently published a map for the Canal Company, embracing the entire west end of the peninsula, from Range 38 W., on which the Iron Range as laid out by Brotherton was indicated.

The excellence of Brotherton's work is shown by the fact that a few changes in the map have been found necessary by the subsequent developments, and that map on a larger scale subsequently published by the Canal Company was of immense service in the development of the Gogebic Iron Range. Brotherton traced the iron formation from the Montreal river, easterly to the center of Range 44. I had the magnetic survey finished by L. E. Paddock, a few years later, when the formation was traced from the center of Range 44 easterly across Range 43, until the magnetic indications ceased near the east line of Range 43.

There are several areas and lines of magnetic attractions which I had traced in the early eighties in various portions of the southeastern part of what is now Gogebic County. The greater part of these magnetic formations remain unexplored.

In the winter of 1880 or 1881, I issued my first option on the Gogebic Iron Range to William Sedgwick, of Ishpeming, who was associated with Captain "Nat" Moore, in exploring property since known as the Colby mine, on the NE¼ of Section 16, T. 47 N., R. 46 W. I issued two options to Mr. Sedgwick, covering the N½ of Section 15, now known as the Tilden mine.

These explorers found the ore on the Colby property, and also that part of the ore deposit on Section 15 adjoining.

The first ore was found by them at what was afterward known as the Nipigon Pit. It is curious to note that the first merchantable ore found on the Gogebic Range was in what was probably the smallest deposit from which ore was shipped, as this pit yielded only about 3,800 tons in all.

In the year 1881, the Cambria Iron Company, of Johnstown, Pa., conducted extensive explorations on the Gogebic Range and in the vicinity of the present city of Bessemer, but early in 1882, this Company abandoned its Gogebic explorations and purchased the greater part of the mines of the Menominee Iron Mining Company on the Menominee Iron Range.

In the spring of 1882, Mr. A. Lanfear Norrie took options and explored lands in Sections 22 and 23, T. 47 N., R. 47 W. in the present City of Ironwood. His work was in charge of Mr. James R. Wood, an explorer of ability and experience. Norrie's first, or summer camp was on the Montreal River, at the place now crossed by the bridge of the Chicago & Northwestern Railway. He later built a log camp in what is now the south part of the City of Ironwood, and this camp was used for several years after the city was built up around it.

My first visit to the Gogebic Range was in the latter part of August and early part of September, 1882. From Ashland we traveled by sail boat, on Lake Superior, to the mouth of Ironton Creek a few miles west of the mouth of Montreal River. The explorers had built a warehouse at this point, and a pony trail from it to the Iron Range, at what is now the Montreal mine; thence along the range to Sunday Lake. Another pony trail was made from the mouth of Black River to the site of the present City of Bessemer.

After spending a few days at the Norrie camp, we moved down to the camps near the southwest corner of section 18, T. 47 N., R. 46 W., where, a day or two later, a messenger brought us samples from the first pit bottomed in merchantable iron ore on the property later known as the Norrie mine. This pit was sunk at the narrowest point in the ore deposit, which was about three miles long. At this point it was only two and one-half feet wide. Subsequent development showed that the ore was anywhere from one hundred to two hundred and fifty feet in width a short distance either side of this pit.

We also spent a few days in a log camp on Section 10. T. 47 N., R. 46 W., on the site of the present City of Bessemer. This camp stood near the spot now occupied by the Soo Line Railway station and from here we visited the explorations which were afterwards developed into the Colby and Tilden mines. Captain Pease, who had had charge of the work accompanied me.

There was an old windfall about a quarter of a mile wide and several miles in length extending from a short distance southwest of the present City of Bessemer northeasterly. This was grown up with raspberry bushes, etc. Near the quarter post, between Sections 15 and 16, T. 47 N., R 46 W., a little patch of ground had been cleared by the explorers and planted with potatoes in 1880 or 1881. Although no work of cultivation or replanting was done on this potato patch, potatoes were dug there every year for at least six years. The snow came early enough to prevent the ground freezing, so the potatoes reproduced themselves year after year.

This was probably the first demonstration of the farming possibilities of that region, and was a forerunner of the many fine farms now to be found in that vicinity.

After visiting all the pits then showing ore on the Gogebic Range, I estimated there were at least 150,000 tons of good ore on the Colby and adjoining properties. Dr. Rominger, the State Geologist, in the same year had made an unsuccessful attempt to reach the Gogebic Range from the east, but only visited the then barren explorations east of Sunday Lake. He afterwards was reported to have stated that he could carry away all the iron ore on the Gogebic Range in his hat. In the following year, Captain Nat Moore and others interested in the explorations, personally conducted him to the range by way of Ashland and showed him the pits in ore, whereupon he became quite enthusiastic over the prospects for iron ore in that district, and withdrew his former estimate of the tonnage.

My first trip to the Gogebic Iron Range occupied two weeks of steady traveling, the greater part of the time on foot.

The first railroad to reach the Gogebic Range was the Milwaukee Lake Shore & Western which was built from the southeast to Watersmeet, Wakefield, Ironwood, etc. In 1884, on my second trip to the Gogebic Range, I proceeded via Appleton, Wisconsin, and to the end of the rails on the Milwaukee Lake Shore & Western Railroad, at the site of the present Village of Watersmeet. From this point I walked over the right-of-way, transit-line, etc., to about the present site of Gogebic Station, then over a trail to the camp of Peter Mitchell in Section 13, T. 46 N., R. 42 W., and from there by trail, boat across Gogebic Lake, and trail to Sunday Lake and the Montreal River.

I again visited the Gogebic Range in 1885, going to the end of the rails, which was then about three miles southeast of the present Village of Wakefield, and thence to the Montreal River, over the right-of-way of the Milwaukee, Lake Shore & Western Railway, afterwards purchased by and now a part of the Chicago & Northwestern Railway. At this time and during the two following years there was great activity and much exploring in the Gogebic district, between the center of Town 47-44, and a point several miles west of the Montreal River.

In the fall of 1885, the rails had reached a point six miles west of the Montreal River, and a trial cargo of ore was shipped from the Colby mine, being the first shipment of iron ore from the Gogebic range.

The railroad was finished to Ashland and ore shipments began in 1886, since which time the shipments of ore from the range have averaged about three million tons per year, amply justifying the first estimates, as to the quantity of ore on the range, made by both Dr. Rominger and myself; he measured this quantity by his hat. I was more optimistic, and estimated it at 150,000 tons.

At the time of my earliest visits, Gogebic Lake probably afforded the best black-bass fishing in the world. This was subsequently ruined by the so-called "fish-hogs," who would catch hundreds of fish in a day and throw them on the bank to rot. This was really a misfortune to the district, for with proper regulation, the magnificent fishing of Gogebic Lake might still have been a prominent feature of the region. There was also good speckled trout fishing, and I know of several "doubles" having been taken in Gogebic Lake near the mouths of streams, one being a bass and the other a speckled trout.

Mesaba Iron Range.

Mesaba is an Indian word, meaning "Height of Land," and originally designated the granite range immediately north of the present Mesaba Iron Range. There are said to be sixteen ways of spelling the name, but I shall use only the one, which seems to me the simplest. I had heard of iron in Minnesota from the time of my earliest acquaintance with the

Lake Superior region, but the information was always qualified by the statement that there was so much titanium in the ore that it could not be worked, and was valueless. However, in 1885 or 1886, some allusions were made to the Mesaba iron district by Minnesota state and the United States Geologists. In reading their reports the statement that the Mesaba formation was the equivalent of the Gogebic formation on the south shore of Lake Superior, attracted my attention, and I then took up the study of that region from such meager data as were available. In 1887, in company with some associates, I located twenty-four thousand acres of land in one entry in the United States Land office. The earliest reports indicated that the Mesaba formation was very flat, dipping only about four degrees to the south. I argued that if this formation was the equivalent of the Gogebic, that the merchantable ore should be on, or near, the footwall and that it would be possible to reach the footwall with a thousand-foot shaft about three miles from the outcrop. The first map of the district published by the U. S. Geological Survey had on it a red line, indicating the junction of quartzite and granite as far as it had been determined. This purported to show the line of this junction from a point several miles east of the Duluth and Iron Range Railroad, to and including the southeast side of what is now known as the "Horn." I therefore, selected all the government land subject to cash entry in a belt three miles wide, the entire length of this red line and lying south of it.

I learned afterwards that the draftsman, in making his map had made a serious blunder, for he started the southwest "swing" of the formation six miles too far east. This threw the west end of my locations nearly three miles off the real course of the range. Thus occurred one of the narrow escapes I have had in achieving great things. However, mines enough were found on the property we purchased to justify the investment, but the results were insignificant compared with what they might have been had the draftsman in the U. S. Geological Survey office, placed the red line where it belonged.

Within a week after my entry of the twenty-four thousand acres, at least one hundred thousand acres of land had been purchased at the U. S. Land Office. These locations embraced the nearest government land left around our locations.

A year or so after this purchase, I made an arrangement with Mr. Russell M. Bennett of Minnesota, and we undertook an extensive exploration of Mesaba iron lands, for which we were to receive an interest in the properties, which were lands owned by lumber companies and which had been purchased for the pine timber on them. This exploration was quite successful, and later led to the development of several of the large mines of that district. My first personal visit to the Mesaba Range was in March, 1891; the trip being made in company with Mr. Bennett and Mr. E. J. Longyear. We followed the tote-road from Mesaba station westerly along the Iron Range, to the exploring camp of Frank Hibbing in Section 14, T. 58 N., R. 20 W., about two miles northeast of the present City of Chisholm, which was the end of the road. From there we went on snow shoes to our own camp on Section 35, T. 58 N., R. 21 W., about two miles northwest of the present City of Hibbing. From here we reached Grand Rapids and the railroad over a tote-road.

At the time Mr. Bennett and I inaugurated our work of exploring there was no merchantable ore known on the Mesaba Range, and we did a large amount of exploring before we found any. Our first "find" was a deposit which was afterwards known as the Stevenson Mine.

My first exploration on the Mesaba Range was on some of the lands I had purchased in my twenty-four-thousand-acre location, and was some of the first work of that kind done by Mr. E. J. Longyear, who subsequently explored a very large portion of the Mesaba Range. He was equipped for this work with a diamond drill, having a rated capacity of six-hundred feet. The site we selected for the first hole was beside the Duluth & Iron Range Railroad, a little more than two miles south from the footwall outcrop, this being the nearest land we had to the footwall. Here it would be easy to obtain fuel for the boiler, the timber of the country having been destroyed many years before by forest fires. A hole was bored at this point about fourteen hundred feet in depth without finding the footwall. After boring a hole fourteen-hundred feet deep with a six-hundred foot drill, we abandoned this site, moving to another place looking for something easier.

In visiting the present Lake Superior iron districts with the early development of which I have been familiar, and now riding over the beautiful roads in an automobile, it is difficult to imagine that these now nearly treeless regions are the same districts I visited only a few years ago, in which there were few or no trails or roads, and covered with a dense forest. Watching this development has been full of absorbing interest and I shall always be glad that I was able to be in it in a way to keep me familiar with its early stages and its later wonderful growth.

EQUIPPING AND SINKING THE NO. 1 SHAFT AT THE HOLMES MINE.

BY LUCIEN EATON, ISHPEMING, MICH.*

In December, 1916, The Cleveland-Cliffs Iron Company decided to sink a shaft and develop the ore known to exist on the SW¼ of the SE ¼ of Sec. 9, T. 47 N., R. 27 W., in the City of Ishpeming, Michigan. This property was named the Holmes mine in honor of the late Jasper Holmes Sheadle, who was vice president of the company. The Holmes mine consists of one forty only, being surrounded on all sides by land belonging to the Oliver Iron Mining Company.

Many difficulties arose in laying out the plant and locating the shaft. The northeast third of the property is a rather high diorite hill, and the remaining two-thirds are underlain by iron formation. The south third of the forty is a swamp. In order to provide a safe location for the shaft, and to make connections with the Lake Superior & Ishpeming Railway, the shaft was located 50 ft. from the north boundary of the property, with its longer axis running northeast and southwest. Five buildings were planned—a shaft house, an engine house, a dry, an office, and a shop building. The shaft house is of steel and the other buildings of light-colored brick. The engine house, office, and dry are located on the hill southeast of the shaft, and the shops are at the foot of the hill east of the shaft. The trestles for stocking ore extend west and south from the shaft house. The surface plan is shown in Fig. 1.

The grading for the railroad tracks and stockpiles required the removal of 35,000 yards of earth. What little earth was not needed for fill was dumped along the north line of the property. The stockpile floors were carefully smoothed and covered with 3-in. plank. Three trestles were built for stocking ore, and one for handling rock. No rock was

*Superintendent, The Cleveland-Cliffs Iron Co., Ishpeming District.

dumped from this trestle, however, while the shaft was being sunk.

On the trestles rock and three grades of ore will be handled by two side-dump cars of 56 cu. ft. capacity each. The cars will run out by gravity and be pulled back by a tail-rope. The cars are fitted with roller bearings.

The shaft house is 150 ft. high, is built entirely of steel,

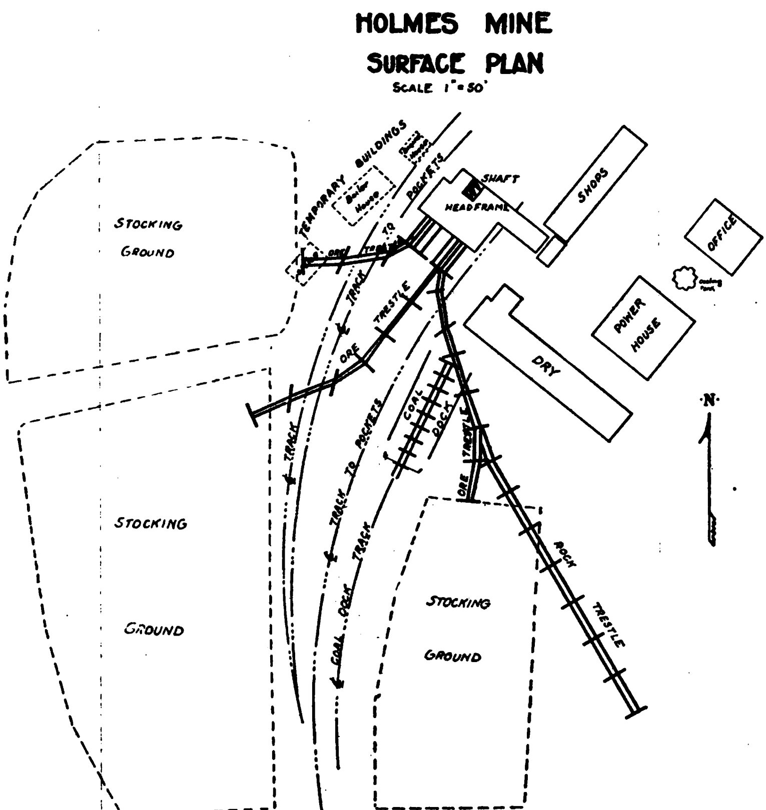
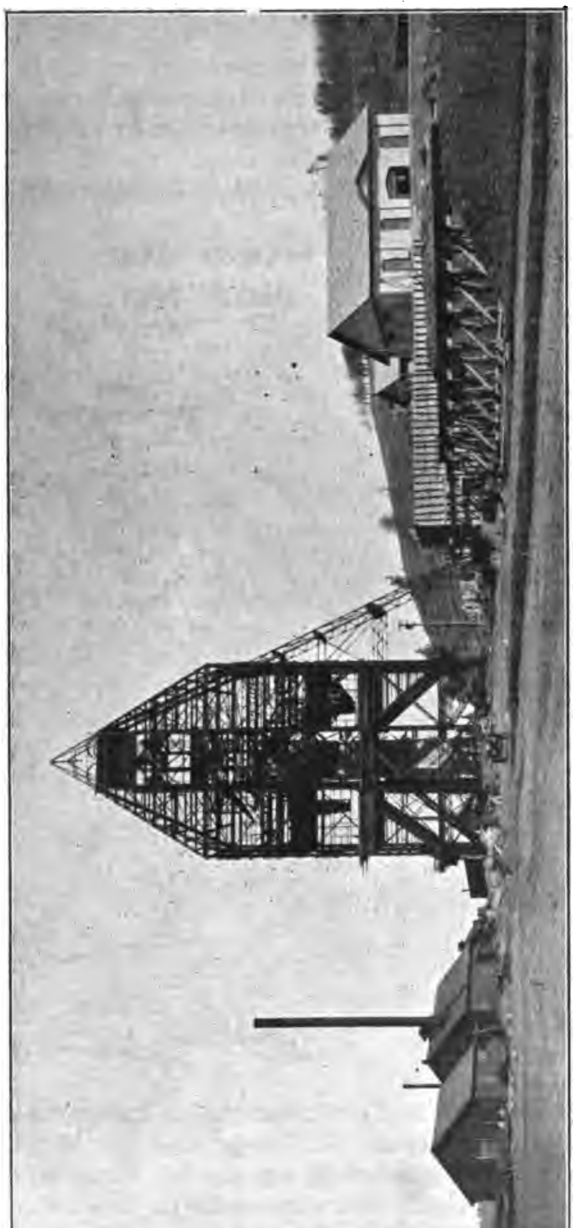
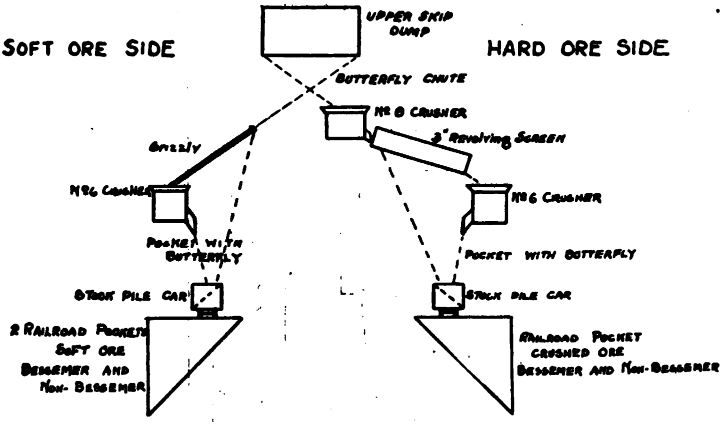


FIG. 1

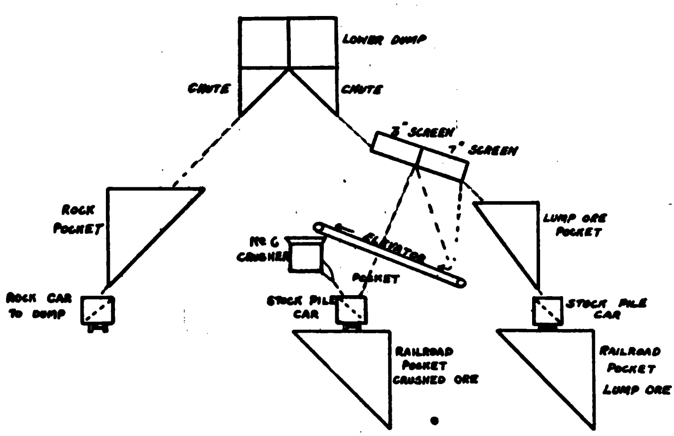
except for the floors and chute and pocket linings, and weighs approximately 340 tons. It has pockets for handling five grades of ore during the shipping season. The landing floor, on which the stockpile cars run, is 40 ft. above the ground level. The shaft house is enclosed above the landing floor, and is painted black. It contains two No. 6 gyratory crush-



SHAFT HOUSE AT THE HOLMES MINE



FLOW SHEET OF CRUSHERS
FIG. 3



FLOW SHEET OF SCREENING PLANT
FIG. 4

HOLMES MINE
SHOP BUILDING
SCALE 1"-4'

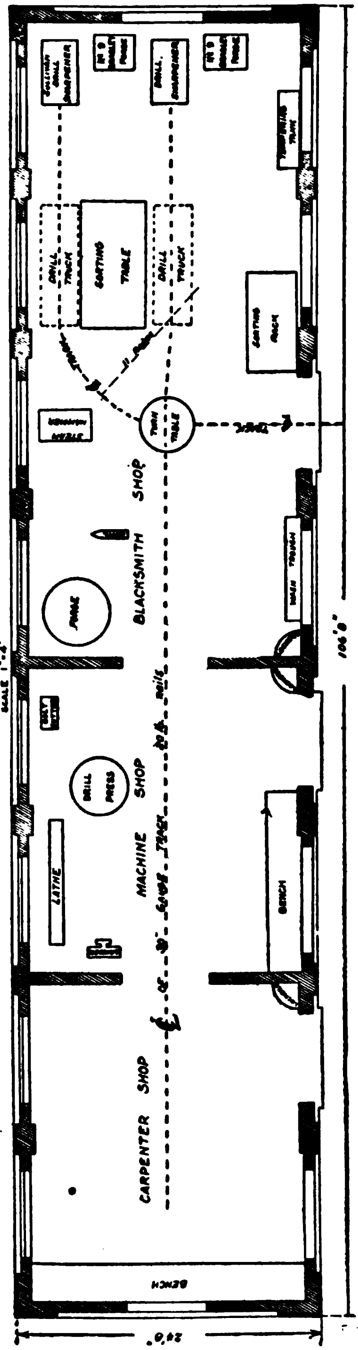


Fig. 5.

ers, which rest on the steel framework, and one No. 8 gyratory crusher, which is supported by a solid concrete pedestal 77½ ft. high. This pedestal contains over 500 cu. yds. of concrete, and has a sub-base 25 ft. square. There are two sets of skip-dumps provided. The upper dumps discharge the ore either directly into the hopper of the No. 8 crusher, which is intended for hard ore, or over a grizzly into the hopper of the north No. 6 crusher, which has grooved concaves for handling soft ore. The ore is diverted either way by a large butterfly valve in the chute below the dump. The discharge from the No. 8 crusher passes through a revolving screen, the oversize from which goes to the south No. 6 crusher. The lower skip-dumps are intended for rock and coarse ore. The rock is handled by the north skip, and the hard ore to be screened by the south skip. The ore to be screened passes through a large revolving screen with openings 7- and 3-in. in diameter. The over-size from the 3-in. holes goes to the 7-in. part of the screen, and the ore passing through the 7-in. holes is elevated to the No. 6 crusher to be crushed. The oversize from the 7-in. holes goes into the lump grade, which is sold to open-hearth furnaces. With both the hard and the soft crushed ore, separation can be made between the Bessemer and non-Bessemer grades. All machinery is driven by electricity, which is delivered at the mine at 2,200 volts, alternating current, 3-phase, 60 cycles. The crushers, screens, elevator, and stocking-cars are all driven by independent motors. The lower skip-dumps, butterflies, and chute-closers are operated by air-cylinders. A view of the shaft house is shown in Fig. 2, and a flow-sheet is shown in Figs. 3 and 4.

The engine house is a brick building 62 ft. square, and contains two electric hoists with 10-ft. drums, built by the Allis-Chalmers Manufacturing Company, a 2,000-cu. ft. air-compressor direct-connected to a synchronous motor, built by the Laidlaw-Dunn Gordon Company, and a rotary transformer for supplying direct current to the electric motors underground.

The office building is also of brick, and is 40 ft. square. It contains the mine office and warehouse, and the captain's office and change-room. It is situated about 50 ft. east of the engine house.

West of the engine house about 30 ft. is the dry, or change house. It is also of brick, and is 160 ft. long and 31 ft. wide.

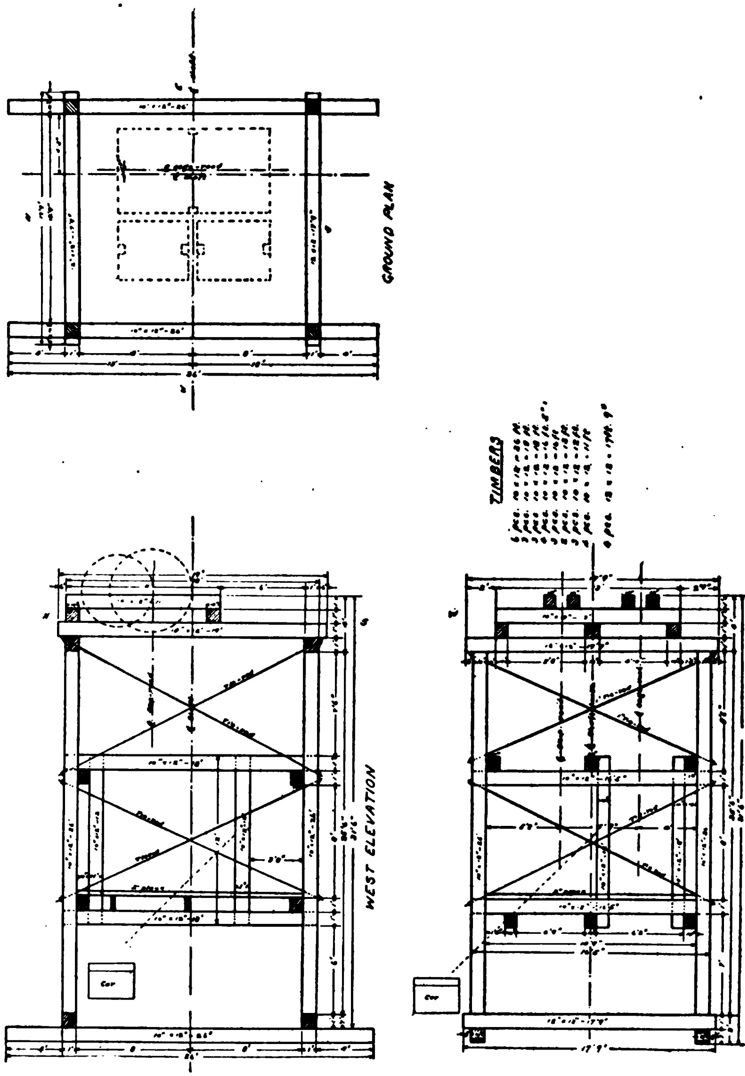
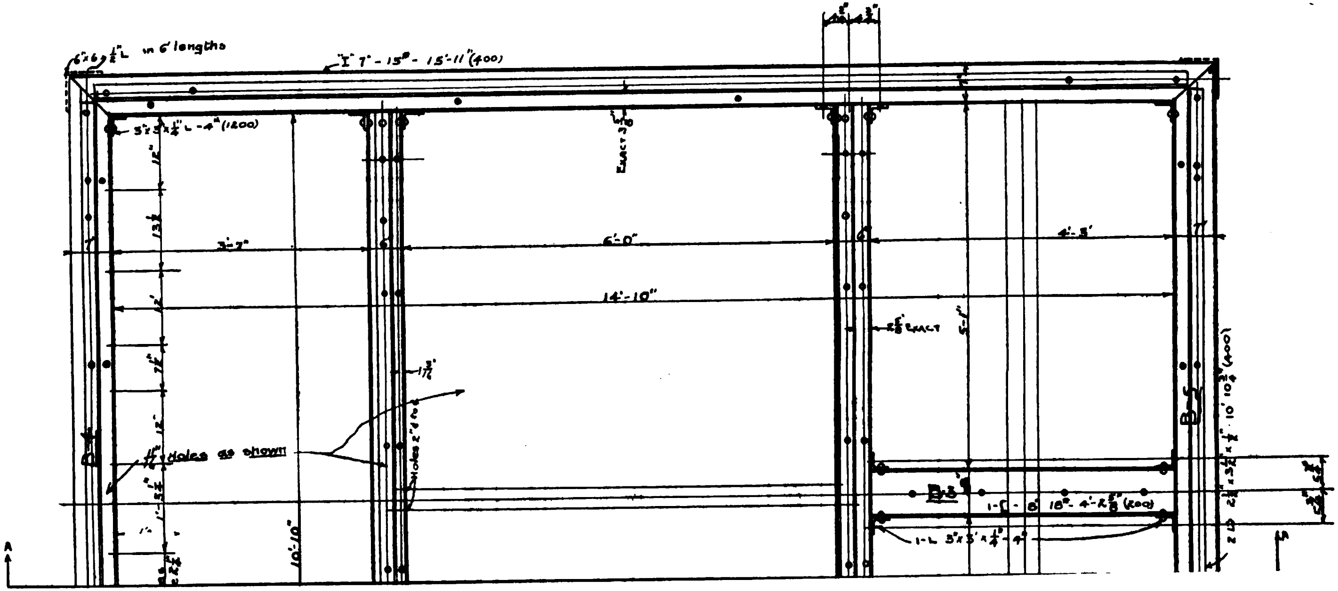
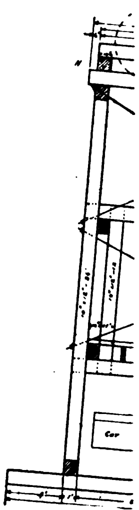


Fig. 6—Shaft House for Sinking Holmes Shaft.



6"x6x½"L in 6' lengths
I 7" - 15# - 15'-11" (400)
5"x3"x¼"L - 4" (1200)
Holes as shown
Holes 2" dia.
3'-7"
6'-0"
4'-5"
14'-10"
10'-10"
5'-1"
Exact
1-[- 8" 18# - 4'-2⅝" (800)
1-L 5"x3"x¼" - 4"
B-3
B-5

II.



It has lockers for 200 men, and is equipped with a toilet, a washroom, and shower baths. The north end of the building has also a hospital room for the care of injured men, and contains a 125 h.p. locomotive boiler, which supplies steam for heating the entire plant.

The shop building is 107 by 25 ft., and is situated at the foot of the hill east of the shaft. It is divided into three

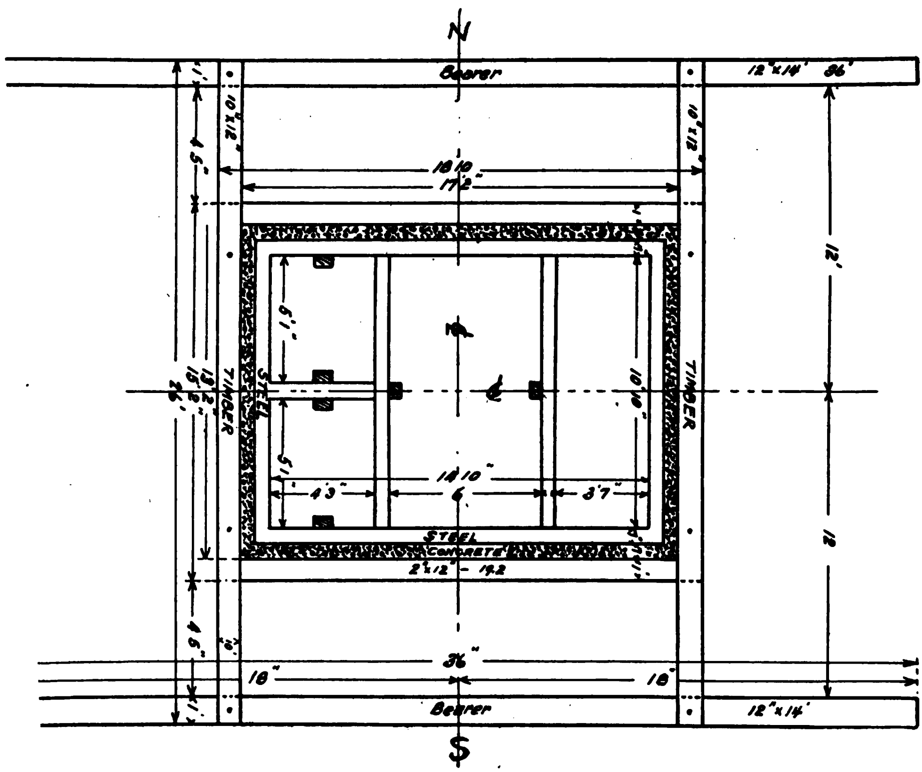


Fig. 7—Holmes Mine, Plan of Shaft.

rooms, the carpenter shop being at the east end, the machine shop in the middle and the blacksmith shop and drill-sharpener shop at the west end. A plan of the shops is shown in Fig. 5.

A small coal dock with a capacity of 500 tons is situated just west of the boiler room. This amount of coal lasts through the winter months. A compartment is provided in the coal dock for Blossburg coal and for coke for the blacksmith and drill forges.

Sinking was started in January, 1916, as soon as the location of the shaft was decided. A temporary office and warehouse, a dry, a boiler house and an engine house were erected north and west of the shaft, where they would not interfere with the permanent buildings. The 125-h.p. loco-

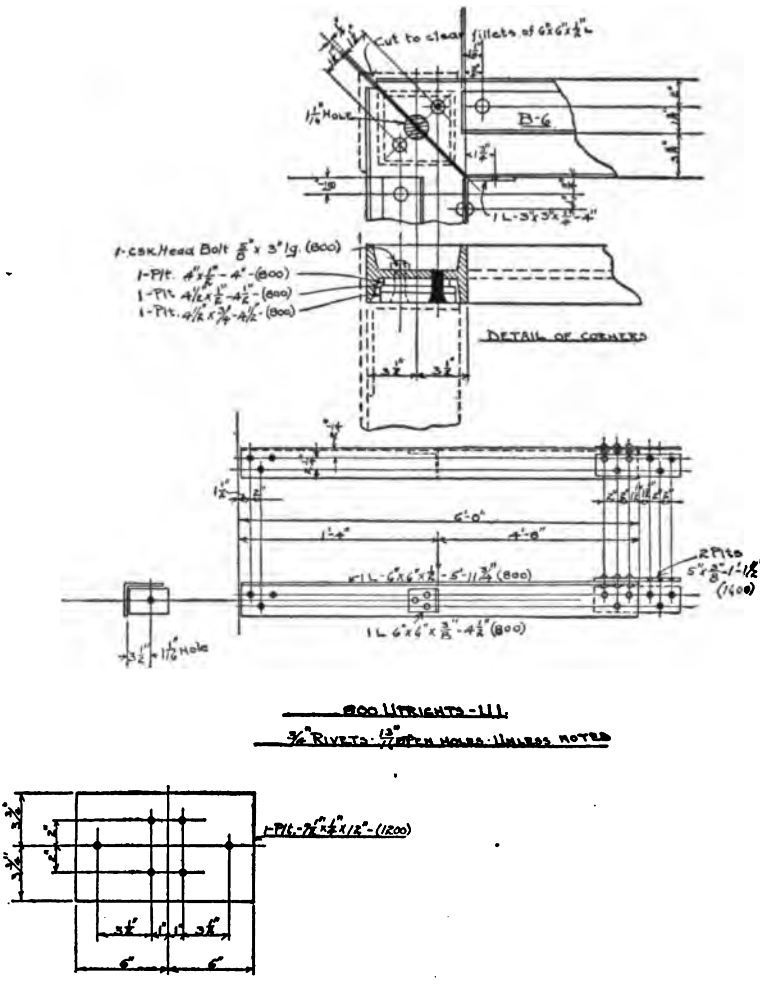


Fig. 8—Steel Shaft Sets, No. 2.

motive boiler, to be used later in the permanent dry, was installed in the boiler house, and two geared steam-hoists with 4-ft. drums were set up on a concrete foundation in the engine house. A 6-in. air line was laid from the Cliffs Shaft mine, a distance of 3,500 ft., and compressed air was trans-

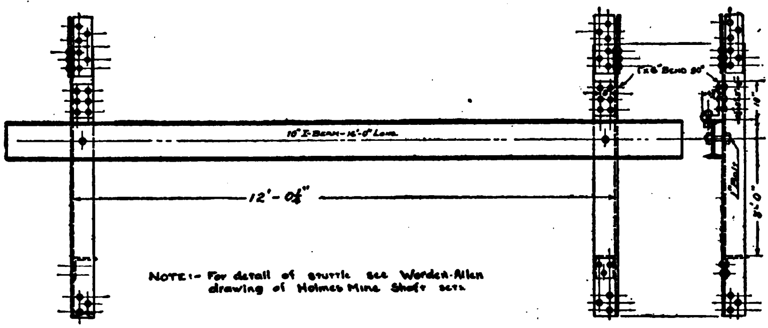


Fig. 9—Steel Shaft Bearers.

mitted through this line until the compressor was started at the Holmes mine in November. A temporary shaft house was built with material on hand, the design being such that it interfered with none of the important members of the steel

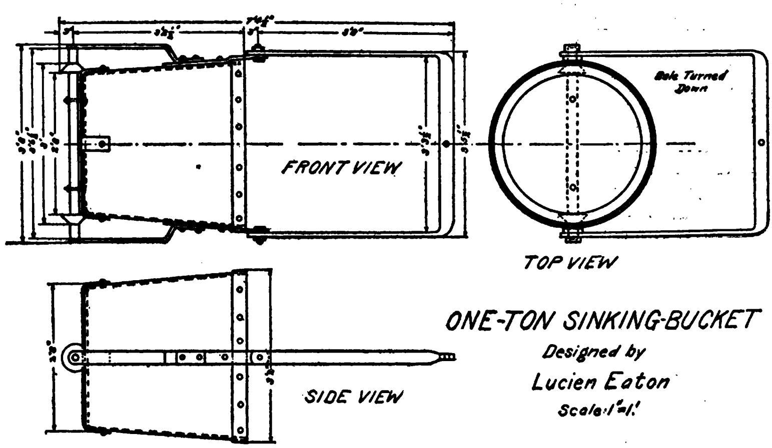


Fig. 10.

shaft house, which was erected during the following summer. The erecting of the permanent shaft house did not interfere with sinking operations. The temporary shaft house was equipped with two chutes for one-ton buckets of special design, and had extra sheaves over the other two compartments of the shaft, for convenience in handling materials. Fig. 6 shows the design of this shaft house.

The shaft is of the standard Cleveland-Cliffs design, 10 ft. 10 in. wide and 14 ft. 10 in. long inside, with two skipways, a cage-road and one compartment for pipes and ladders. The sets are made of 7-in. standard I-beams, reinforced with angle-irons. The lathe is of 3-in. tamarack plank, and the casing between compartments is of 2-in. tamarack plank. Split-cedar lagging is used for blocking behind the lathe. The studdles

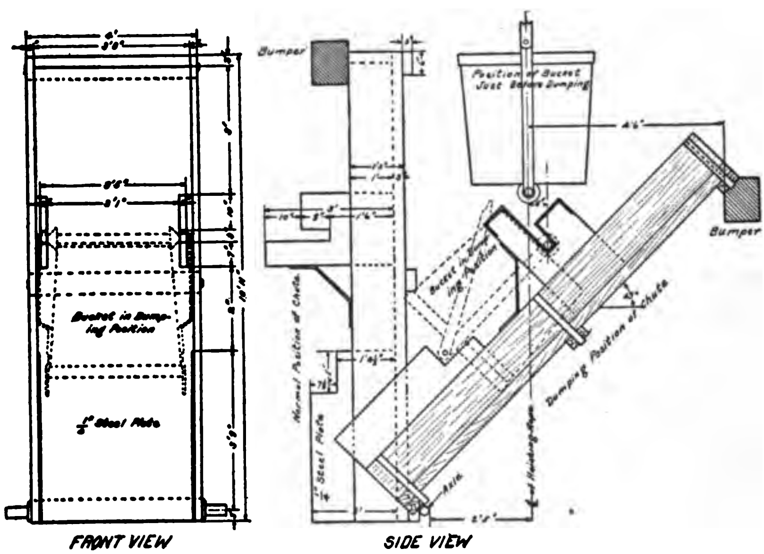


Fig. 11—Dumping Chute for One-Ton Sinking Bucket. Designed by Lucien Eaton.

(Note:—Chute is pulled down to dumping position by an air-lift, and is raised by a counter-weight.)

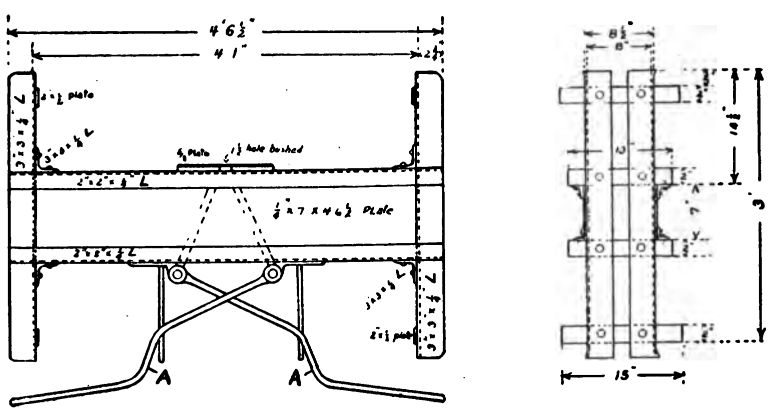
are made of 6 by 6 by ½-in. angle iron. Ten-inch I-beams 16 ft. long were used for bearers at approximately 100-ft. intervals. These were wedged into place with steel wedges in hitches cut in the sides of the shaft, and the hitches were then filled with rich concrete. As the shaft was started in gravel, it was sunk large enough to allow concrete to be poured outside the steel shaft-sets until the ledge was reached. The shaft was concreted to a depth of 52 ft., and then narrowed to fit the steel sets closely. The plan of the shaft collar and steel sets, and the method of attaching the bearers are shown in Figs. 7, 8 and 9.

The sinking-buckets were specially designed so as to dump easily and safely, eliminating, as far as possible, the dangers and inconveniences of the ordinary type of bucket used in this district, which is hung below the center of gravity and is held in a vertical position by slip-rings on the bail and pins on the rims. The cross-heads were also of special design, because of the very slight clearance between the dump and sheave. Views of these buckets and cross-heads are shown in Figs. 10, 11 and 12.

The drills used in sinking were of the Jackhammer type,

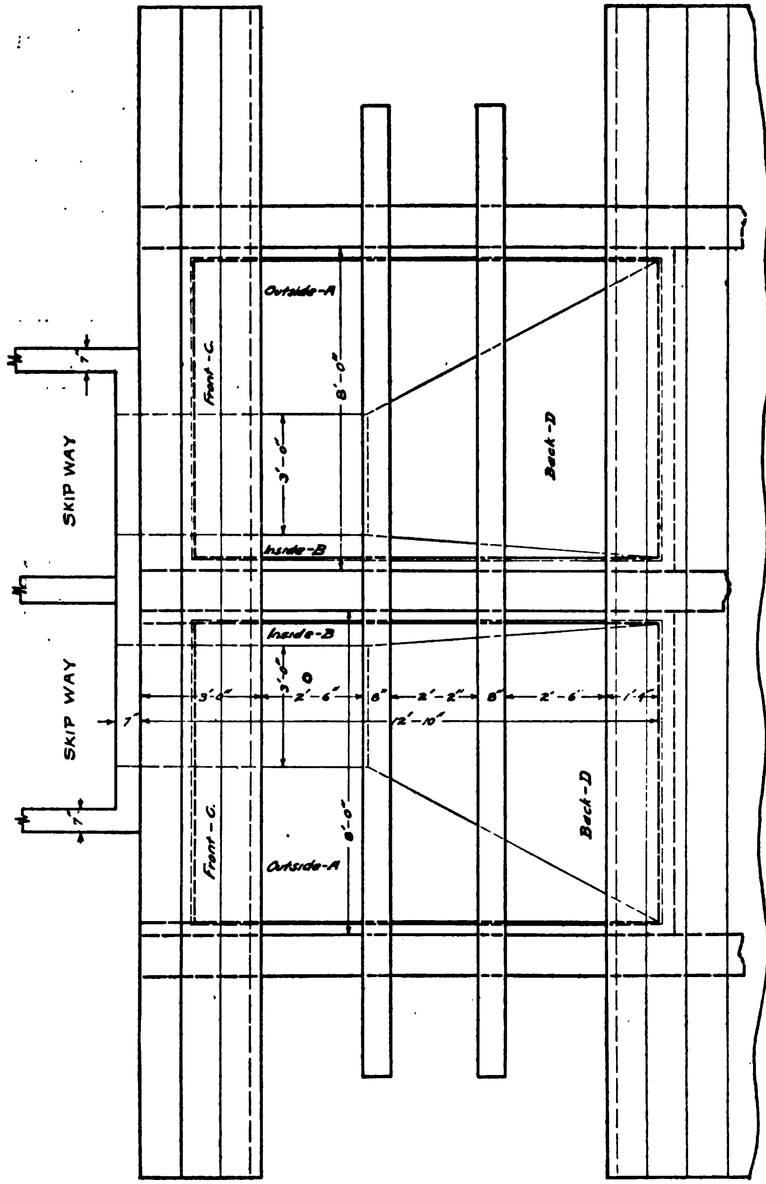
CROSS-HEAD FOR SKIP-ROADS

FIG 12



A-A = 2 Levers for operating Jaws that hold Clevis to cross-head, when hoisting. one on each side

both the Sullivan DP33 and Ingersoll BCR430 drills being used with ⅞-in. hexagon hollow steel and Carr bits. Ten machines were used, all being connected by 25-ft. lengths of 1-in. hose to a special manifold. The manifold was connected to the 3-in. air line by a 35-ft. length of 3-in. hose, this one connection being all that was required before drilling was started. In present practice the manifold is hung under one bucket during drilling time, and is hoisted directly to the shaft house with all the drills attached when the round is finished. Blasting was done by electricity, a No. 5 blasting bat-



LAYOUT OF UNDERGROUND POCKET
FOR
HOLMES MINE
CLEVELAND CLIFFS IRON CO.
ISHPEMING MICHIGAN

Scale ⅛" = 1' LKS October 30, 1916.

Fig. 13—No. 1.

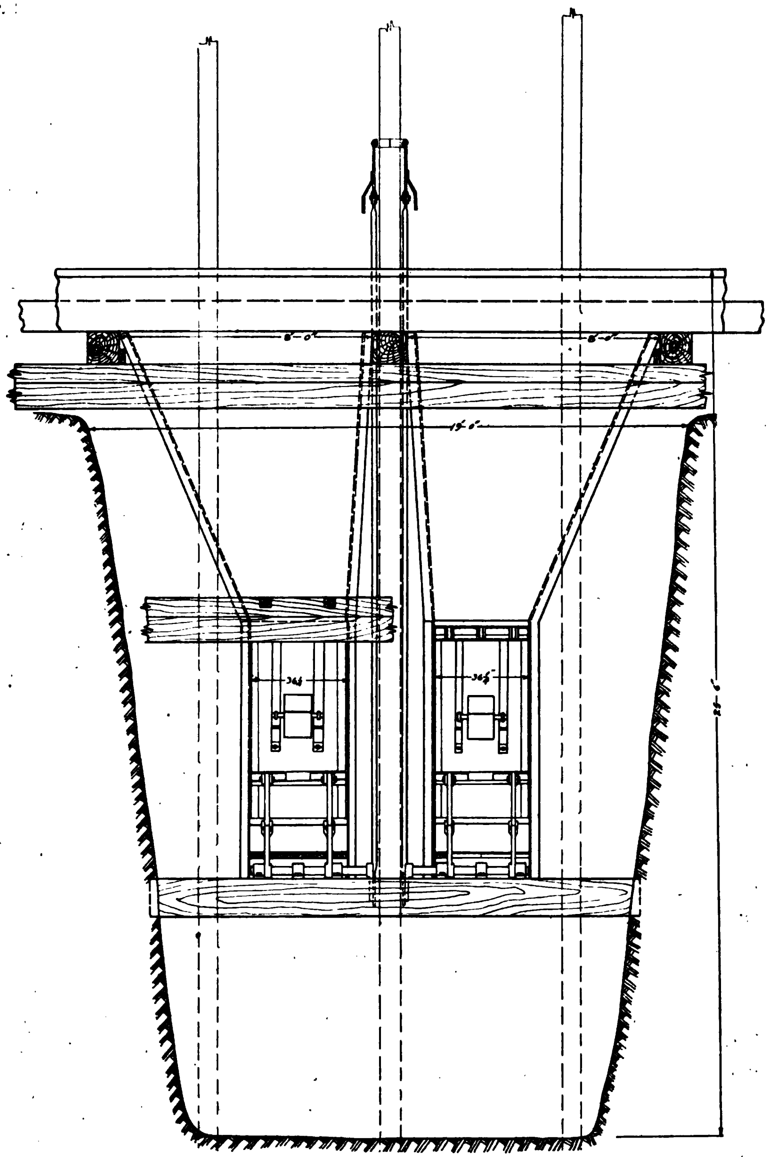


Fig. 13—Layout of Underground Pocket, No. 2.

bonus, or $4.90 a day. The bosses receive 25c a day more than the miners. This rate of pay was increased 10 per cent. on December 15, 1916. Setting bearers is considered part of the regular work, but cutting pump houses and plats is not so considered, and allowance is made for this work in calculating the bonus.

The progress made in the shaft is shown in the following table:

Date. 1916.	Depth Sunk, Feet.	Depth, Feet.	Remarks.
Jan. 11		0	Started work.
Feb. 1	24	24	Built temporary shaft house.
March 1	43	67	Put in steel sets and concrete.
April 1	92	159	Three shifts started March 1st.
May 1	89	248	Cut pump house at 210 feet.
June 1	123	371	
July 1	100	471	Cut pump house at 455 feet.
Aug. 1	121	592	
Sept. 1	120	712	
Oct. 1	108	820	Cut pump house at 740 feet.
Nov. 1	147	967	Reached first level at 963 feet.
Dec. 1	58	1,025	Cut plat and pockets at first level.
1917.			
Jan. 1	57	1,082	Cut plat and pockets at second level at 1,063 feet.
Jan. 12	34	1,116	Finished skip-pit.

The progress made in October, 147 feet, is the record for the Marquette Range.

In order to hasten the development of the mine, arrangements were made with the Oliver Iron Mining Company to continue to the Holmes shaft a drift which had been driven on the 16th level of the Section 16 mine across the Holmes mine boundary for a distance of about 1,000 feet. This drift was turned to the north and had been driven 836 feet in rock by the 13th of January, 1917, at which time work was stopped. Connection was made from the shaft on January 29. The 2nd level of the Holmes mine and the 16th level of the Section 16 mine are at the same elevation. This connection between the two mines will greatly improve the ventilation in both mines, and afford a safe second outlet in case of accident.

Figs. 13 and 14 show the underground pocket and the plan of the plats.

William Tamblin was mining captain in charge of the shaft, and the shift bosses were Fred Prudom, Wm. Olds and Jonas Johnson.

SHAFT PLAT FOR HOLMES MINE

SCALE 1" = 10'

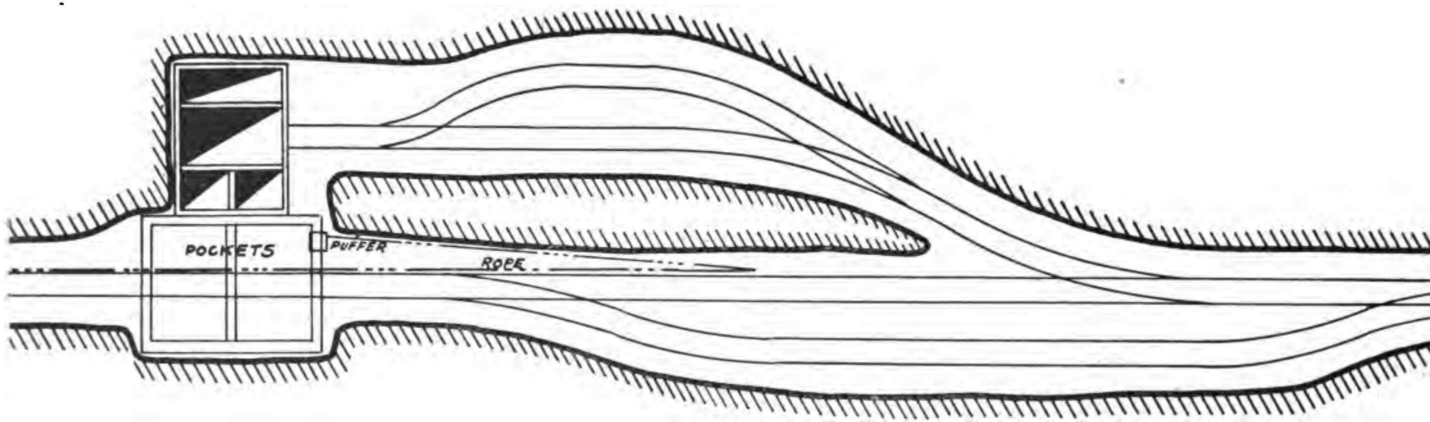


FIG. 14

To provide a suitable timber yard, five acres of land immediately north of the shaft were leased from the Oliver Iron Mining Company, and a tunnel was driven from this land into the sidehill, a distance of 200 ft., reaching the shaft 30 ft. below the collar. Mine timber is unloaded from railroad cars on the track north of the shaft, and rolled down the 30-ft. embankment to the timber yard, the elevation of the track making it possible to have piles about 35 ft. high. The timber will be drawn from the lower side of the piles, framed, and trammed to the shaft through the tunnel. The timber trucks will be run on the cage, taken to the proper level, and hauled to the point of unloading by electric motor.

CRUSHING PLANT AT BRIER HILL SHAFT.

BY FLOYD L. BURR, VULCAN, MICH.*

Of late years furnace men have come to require that the ore furnished them shall have been passed through a crusher, and in consequence there are now many shafts in our Lake Superior iron ore region which are equipped with crushing and screening machinery. On account of this demand of the furnace men, it was decided, early in the year 1915, to install a crusher at the Brier Hill shaft of the Penn Iron Mining Company's Vulcan mines.

As often is the case in such affairs before any study of the situation is begun, the proposition was assumed to be one of great simplicity and very moderate cost. A very little detailed study disposed of this idea, and it was realized that the matter was of considerable complexity and would involve a somewhat elaborate construction and correspondingly considerable expense.

To understand the conditions one should take careful note of the relative location, both horizontally and vertically, of the shaft, its concrete lining, the various hoisting cables, the hoisting house, the skip dump, the battery of shipping bins or pockets, the side tracks serving the pockets, the main line tracks and right-of-way of the C. & N. W. railway, the stockpile ground, the tramming machinery, etc. The proximity of the shaft to tracks and the lack of any unused headroom should also be noted. The illustrations herewith may help to convey the idea of the congestion at this spot.

This shaft is circular, 14 ft. in diameter and lined with concrete. Down to the rock surface, some 60 ft., this lining is massive, being 20 ft. square outside. The shaft is surmounted by a heavy steel headframe, the principal portion of which is only 16 ft. 3 in. wide; the vertical columns resting upon the concrete lining just mentioned. The structure is 94 ft. high up to principal headsheaves. On account of the

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*Structural Engineer, Penn Iron Mining Company.

large ratio of height to base width, there is quite a marked sway when the skip is dumped. The landing platform is 43 ft. above the ground and is formed by extension of the head-frame to the south. A trestle leads from this platform east to the pockets and west to the stockpile. On this platform was the lander's house, where the electric tramming machinery was controlled, and on the ground under this platform is a small house where the tramming machinery is situated. At

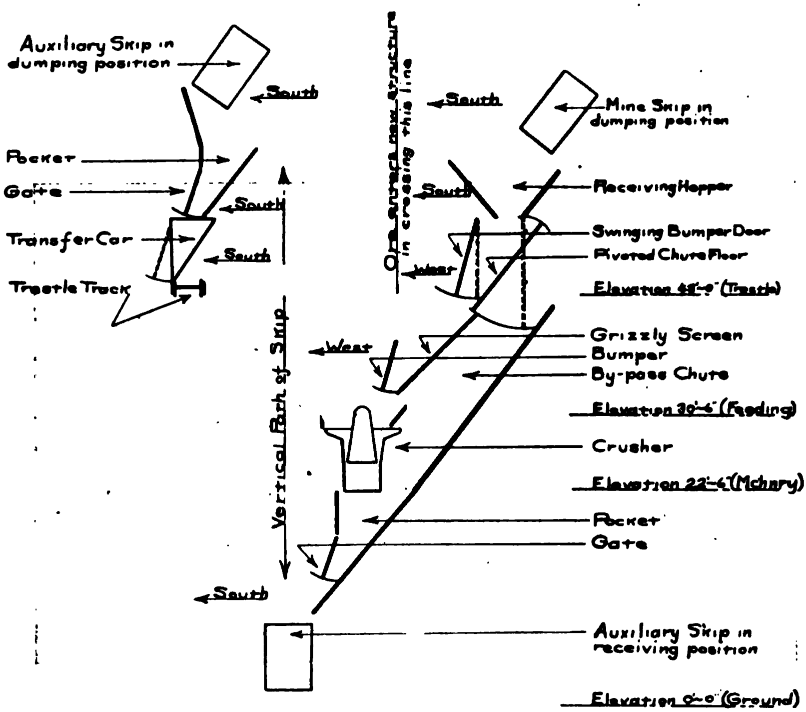
BRIER HILL SHAFT

only clearance distance still farther to the south are the rail-road tracks which serve the pockets, and immediately beyond them the main line of the C. & N. W. railway.

The preliminary study showed that unless the mine skip could be made to dump at a higher elevation, the ore would have to be elevated by some auxiliary machinery either before or after passing through the crusher. It was proposed to build up an extension of the headframe to carry the skip head-sheave at a higher elevation, thus allowing the dump to be raised and the crusher to be placed so as to receive the ore direct from the mine skip dump and yet discharge it at an elevation above the tram car. This proposition, however, was soon dismissed as impractical on account of the resulting instability of the headframe, the cost, the delay to production which would be entailed by the changes in the steel work of the headframe structure, and the difficulty of getting suitable foundation and building a stable structure to support the crusher at a height of more than 50 ft. above the ground.

BRIER HILL SHAFT

Many schemes were proposed and followed out in study. The first scheme studied was essentially an arrangement whereby the ore would be dumped, as in the past, from the mine skip, fed into the crusher, and after passing the crusher be allowed to flow into a new auxiliary skip which would then be hoisted vertically and dumped at same elevation as mine skip dump, thereafter being handled, as was the practice, by tram car on the trestle to shipping pocket or stock-pile. Following that there came a succession of plans involving bucket elevators, belt conveyor elevators, and many other contrivances too wonderful, as well as too numerous, to mention. But after a while the cycle was completed and we had arrived again at the idea of an auxiliary skip conveying the unit of crushed ore vertically upward to a sufficient elevation to permit discharge into the usual tram car (locally called "transfer car") on the trestle. There was also much discussion and study having to do with the necessity of and arrangement for screening the ore before feeding into the crush-



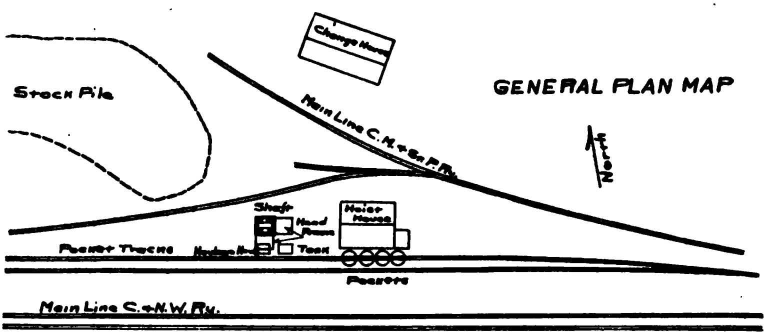
RECTIFIED PROFILE
showing
TRAVEL of ORE

er, consideration being given to stationary bar grizzly, traveling grizzly, revolving grizzly and other forms of machinery. The stationary bar grizzly was finally adopted.

The layout provides for dumping the ore from the mine skip into a short chute from which it flows on over some 10 ft. length of grizzly set on an inclination of 45° from the horizontal and then into a No. 7½ Telsmith Breaker (gyratory crusher), the crushed ore thence falling vertically into a storage pocket from which it may slide, at the will of the operator, into an auxiliary skip. The skip is then hoisted and dumped into a second storage pocket from which it is finally allowed to flow into the waiting transfer car.

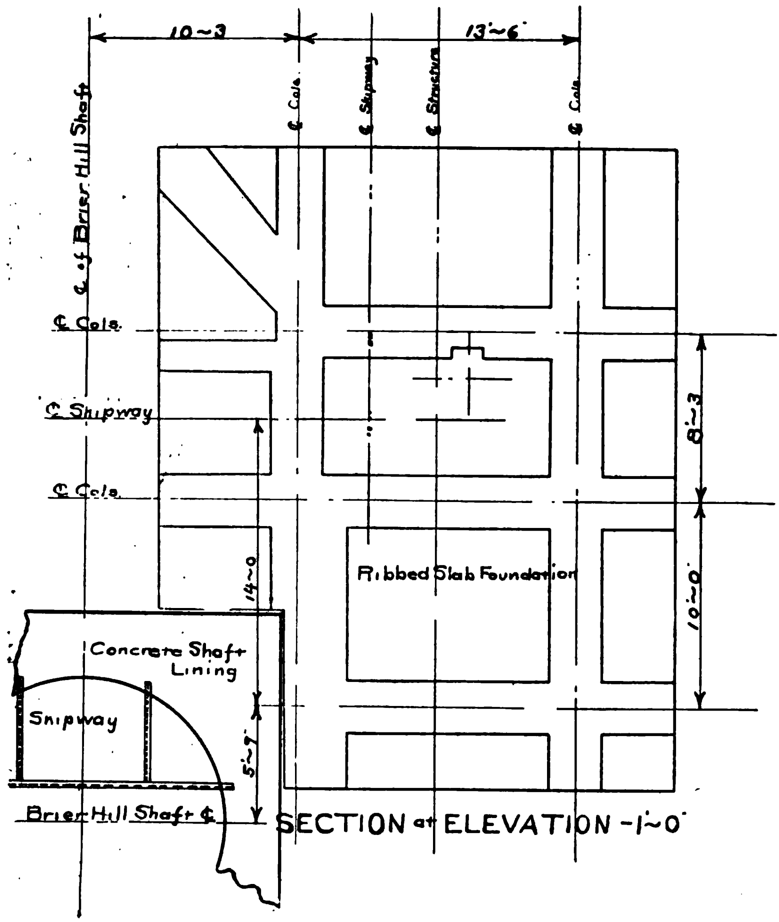
The floor of the short chute mentioned above is so arranged that it can, upon occasion, be swung on a horizontal axle into a vertical position, thereby allowing the falling ore to drop into a by-pass chute below the grizzly whence it enters the pocket below the crusher and escapes screening and crushing. In ordinary operation this same by-pass chute receives the fines which pass through the grizzly and conveys them to the pocket below the crusher.

The idea is to handle a 6-ton mine skip load of ore as a unit, and accordingly each skip load passes separately through crusher, pocket, skip, pocket, and car to stockpile or shipping bin. This arrangement provides for various grades of ore.



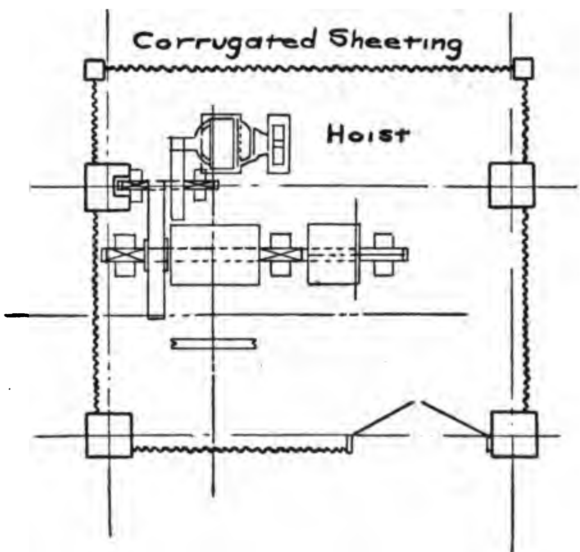
Since the mine skip has a capacity of 6 tons, the auxiliary skip and the two pockets must likewise hold 6 tons. They are actually built so as to hold a liberal overload.

The skip is of the usual Kimberly type. It is quite heavy, weighing 7,220 pounds. In place of the usual bail clevis and thimble, it has at the top two 24 in. sheave wheels around which pass lines of 5/8 in. flexible steel hoisting cable. It is counterbalanced, or overbalanced, by a balance weight traveling beside it. The weight of this is 13,220 pounds. The unbalanced portion of the load is hoisted and lowered by means of a self-contained electric hoist of decidedly novel design, situated vertically above the skip and balance in a small engine room at the top of the structure, that is about 90 ft. above the ground. This hoist is an original design of F. H, Armstrong, mechanical engineer of the Penn Iron Mining Company. The use of this type of hoist in place of the standard drum-on-the-ground type makes possible the slender towering structure, as it does away entirely with the need of backstays. Early in the study some form of geared hydraulic



cylinder hoist was very carefully considered. The hoist is automatic, and the operator has only to throw his switch lever over in one direction to cause the skip to rise, dump its load and stop, and in the other direction to cause it to lower and stop in receiving positoin at the mouth of the lower pocket. The skip travels about 70 ft.

The two pockets are provided with heavy steel gates, or doors, hinged at the top and opened and closed by a system of toggle bars. The opening of the gate is done largely by the lateral pressure of the ore, but it is necessary to start the opening with the lever. The closing is done, of course, against an empty pocket, otherwise this type of gate could not be used. By means of small flexible steel cables, the lower pocket door is operated normally by one man, who is stationed in a small room on the trestle level and who has control there of grade signal bells rung from underground, the dumping ot the mine skip, the starting and stopping of the crusher motor,

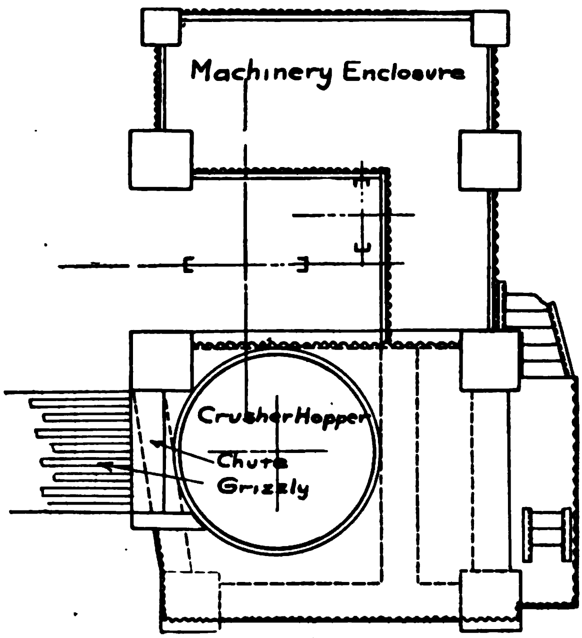


Section at Elevation +93′—0″.

the filling of the auxiliary skip by operation of the lower pocket gate, the hoisting, the filling of the transfer car by operating the gate to the upper pocket, and finally, through electrical control of the haulage engine, the travel of the transfer car out over the trestle to stockpile or shipping pockets. Of course, he is provided with a telephone for any desired communication with mine or engine house.

The structure is of concrete, reinforced largely with old steel hoisting rope. It is founded at an elevation of 5 ft. below the ground surface and extends to an elevation of 96½ ft. above ground, having thus a total height of over 100 feet.

The foundation conditions were far from ideal on account of fine sand and filled ground surrounding the shaft, which is immediately adjacent. The structural foundation is a ribbed slab, consisting of a slab proper of 27 in. in thickness, surmounted by and cast monolithically with a system of intersecting ribs, 30 in. wide and 33 in. high. It forms a "raft foundation" 25 ft. wide by 31 ft. long, minus a corner 6 ft. by 9 ft., where the shaft foundation extends into the rectangle. The foundation contains about 93 cu. yds. of 1:2:4



Section at Elevation +33'—0".

concrete, or 30% of the 312 cu. yds. of the whole structure, and was cast all in one day on June 29th, 1915.

Up to an elevation of 51 ft. the columns are 30 in. square, and above that they are 24 in. and finally 18 inches. They are not true reinforced concrete columns, since the flexible steel cables are not capable of assisting the concrete to resist compression. The reinforcement is there to resist tension, due to bending strains from wind, crusher vibration, etc. The girts are 18 in. wide and 48 in. deep where not influenced by special considerations. Floor and roof slabs are generally 6 in. and 8 in. thick. Steel channels, projecting out of the concrete as cantilevered beams, serve to support stairway flights and thin concrete landing floor slabs. The machinery rooms are enclosed by galvanized corrugated steel sheathing curtain walls, on a wooden framework between columns. A steel railing guards all exposed floors and stairs. Below the crusher level solid 6-in. and 10-in. reinforced concrete curtain walls stiffen the structural framework and incidentally form enclosures. Thus a very useful storage and general purpose room is produced at the ground level. Attached to the southwest column, at an elevation well above the trestle, is a steel swinging crane used for handling the transfer car and for general purposes. Situated vertically over the crusher is a concrete cantilever which serves as a crane for handling the crusher parts. Also being situated opposite the cantilever work which constitutes the upper pocket, it acts as a balance thereto, keeping the center of gravity more nearly in the center of the structure than it would be otherwise. The hoisting room at the top of the structure is entered by means of a short stairway from the sheave floor of the older steel headframe.

The construction and equipment of the plant was done entirely by the mining company's force. Excavation was begun June 7, 1915, and the ore started to go through the crusher November 15, 1915. The total cost of the plant on the books was about $14,300, of which $5,850 was for purchase and installation of machinery, and $2,200 for miscellaneous steel work, while the rest included excavation, grading, the concrete structure, electric light wiring, foremanship, changes in adjacent trestle, etc. It should also be recorded that there was a considerable quantity of steel used which for certain bookkeeping reasons was not charged to the job, so that including all items the actual cost may have run close to $15,000.

The concrete used was mostly of the proportion 1:2:4, but there was some 1½:2:4 used for special reasons. Some of this extra rich concrete was used in the upper portion of the structure, when the weather had become cool and it was desirable to promote rapid setting in order to crowd the date of completion and commission. The reinforcement was mostly old steel rope, but there was also used some old rails and some "Slick" deformed bars. A carload of old rope was purchased for the job.

The concrete skeleton structure was cast in 17 parts, or on 17 days—June 29, July 26, July 27, July 29, July 30, July 31, August 14, August 18, August 21, September 1, September 4, September 15, September 17, September 24, September 25, September 28, September 29. The curtain walls were cast August 10, 11 and 12. The mixer used mixed a charge of 2 sacks cement, 4 cu. ft. sand and 8 cu. ft. of rock, turning out a product of about 9 cu. ft. of mixed concrete. This concrete was hoisted by electric winch in an automatically dumping bucket traveling in a wooden tower, and distributed to the forms by means of wooden launders or chutes. The tower was about 112 ft. high and was rendered stable by braces to the adjacent steel headframe. Crushed rock was produced by a portable crushing and screening outfit working on the waste rock pile at the Curry shaft, about 2,000 ft. away. Sand was hauled from a pit only 300 ft. distant.

The construction and equipment of this plant was carried to completion with absolutely no interference with ore production. This condition was specified originally and influenced the design to some extent.

The operation of the plant has been successful. In summer weather one man can easily operate it alone. In winter there is considerable trouble by reason of ore freezing and accumulating on the grizzly and in the various chutes. Then two men are needed. There have been some stoppages of the crusher due to the congealing of the lubricating oil in cold weather. This oil is circulated around the crusher shaft by means of a small pump built integral with the crusher.

To those who are operating crushers there will probably occur this question, "What about vibration?" The crushing machinery produces two effects, both of which may be termed vibration. The first is a rapid vibration or jar which is noticeable by placing the hand against one of the concrete columns; the second is a slower vibration, which the writer prefers to call gyratory swaying. The latter is most noticeable, of course, at the top of the structure. This swaying can be easily seen and felt. It seems to be most pronounced when the crusher is running empty. When the crusher receives a load of ore the rhythm seems to be partially broken, probably due to interference and partial neutralization of vibration waves.

NOTES ON THE CALUMET & HECLA MINE FIRE.

BY JOHN KNOX, JR., CALUMET, MICH.*

This fire is worthy of note only on account of the conditions existing in connection with it, viz:

1. The fire was 4,000 feet from surface and comparatively close to a producing shaft from which our main pumps were operated.

2. The shaft pillars were crushed to such an extent that the gases had free passage through the crevices around the fire walls.

3. It was very important to maintain production from levels below the fire area without interfering with the fire fighting operations.

4. The air current in the shaft carried off the smoke and gases so that it was possible to establish a fresh air base on the opposite side of the shaft.

On January 8th, 1916, about 9:30 p. m., fire was reported in a partially caved stope about 250 feet south of No. 5 shaft on the 41st level; this stope was connected with other caved stopes above and below. There were 400 men working in this part of the mine at the time the general underground fire alarm was turned in, but within thirty-five minutes all of them were reported as checked out. The general alarm system is perhaps worthy of note. It consists of large bleeders on the air mains; when opened they cause a sudden loss of pressure at all working faces. The use of the bleeders can be readily distinguished from the gradual drop of pressure due to shut down of the compressors; hence there are no false alarms. A fire drill is held every sixty days, the men being required to report at the plats ready to go to surface through a shaft other than the one to which they are accustomed. In this way there is no confusion, as each man is familiar with the various routes all of which are marked by sign boards. Each level is equipped with a fire wall and door located in the shaft pillar, and a barrel of soft clay for sealing is kept nearby. Formerly these fire walls consisted of brick or con-

*General Superintendent, Calumet & Hecla Mining Co.

crete frames with steel doors; but, due to pressure, it has become impossible to maintain masonry walls and present practice is to build wood block walls about 24 in. thick from which the steel doors are suspended; as the blocks crush, the location of the hinges can be moved to insure closing of the door. This was the type of wall in use on the level affected by the fire.

As soon as the mine rescue crews arrived on the ground, hose and pyrene extinguishers were used in an attempt to extinguish the fire; but the smoke became so thick that the lamps could not penetrate the fog and it was impossible to direct the streams. As the timber burned out and caving occurred, the operation became dangerous and it was decided to cut off the area by means of the doors located in the shaft pillar. The fire doors on each level above the 41st were closed and clay applied to the joints, but the next day it became evident that the gases were coming through crevices in the rock thirty feet outside of the fire doors and the side walls were plastered with clay in an effort to stop this leakage. As the temperature increased, the doors from the 38th to the 41st levels became so hot that clay would no longer adhere, so it became a question of building a gas tight wall on each of these levels sufficiently strong to withstand the crushing pressure. Square timber blocks about forty-eight inches long laid in cement were selected as being the most advantageous material because the units could be prepared on surface and sent down as required. Before building these walls it was necessary to pick up three feet of crushed bottom and lay a bed of cement and clay, all of which had to be done by men using breathing apparatus. As it was apparent the six sets of Draeger apparatus of the 1910 type owned by the Calumet & Hecla Mining Company, were inadequate for the service; and as the Bureau of Mines car was in Iowa at the time and could not reach Calumet before January 14th, none of their apparatus was available, six sets of Fleuss apparatus were borrowed from the Copper Range and Quincy Mining Companies together with five sets of Draeger apparatus of the 1914 type from the Cleveland-Cliffs Iron Company. Mr. C. S. Stevenson and Mr. J. H. Williams of the latter company, experts in the use and care of breathing apparatus, volunteered their services, and much of the success of our operations was due to their efforts. The Calumet & Hecla apparatus, which had been in use continuously from Saturday until Wednesday, was much in need of repairs; in fact, there were so many minor derangements that it would have been impossible to continue its use had the distance from the fresh air base been over 150 feet. Fortunately the Calumet & Hecla had seventy-five helmet men holding Government certificates, and eighty-five more partially trained in the use of breathing apparatus, so that there was no shortage of helmet men. As fifteen minutes seemed to be about as much as a man could stand at one time, it was arranged to have fifteen minute working periods and thirty minute resting periods, the men changing in place; from three to four men were employed on a wall at one time.

On the 41st level during the building of the new wall, which was located about three feet outside of the regular fire door, the cumulative effect of the heat and pressure caused the old wall to collapse exposing the helmet men to the gases and direct heat of the fire; after a temporary stopping had been effected, it was decided to face the new wall with cement simply as a protection against the heat. The quickest way being the best, the cement was wet in the sacks and laid in against the inner face of the wall and allowed to set. This protection proved effective and needed no attention during the five weeks required to smother the fire.

The following notes bring out the important points as observed by the men in charge of the various operations during the fire.

1. The conditions at this fire were ideal for the successful use of mine rescue apparatus in that it was never necessary to penetrate the zone of irrespirable gases for a distance of more than 150 feet. It was necessary, in several instances, for the wearers of machines to retreat to fresh air on account of some disarrangement of the machines yet this was accomplished in every instance without any serious results. It was evident, however, that should it have been necessary to penetrate the gases for a much greater distance, the lives of the rescue men would have been endangered in several instances. It would seem, therefore, from the experience gained at this fire, that while rescue apparatus may be of the greatest value in the saving of life and property when used within its limitations, yet these limitations are far more narrow than is commonly supposed by those who lack experience in their use under extreme conditions.

2. Another lesson of value brought out by the fire was the necessity of having at the point where the rescue apparatus is in operation a man of unusual ability in the adjustments, repair and operation of the machines. It should not necessarily be the duty of this man to actually participate in the manual operations incident to the rescue operations, except in extreme cases, but rather to quickly remedy any troubles which may arise in the use of the machines, thus avoiding the necessity of removing machines to the surface for trivial repairs and adjustments. In furtherance of the work of this man it is of the greatest importance that all machines which are sent to the surface for major repairs, or for recharging shall be returned to him in perfect working order and fully charged. This involves the necessity of having a man on surface who is an expert in the repair and maintenance of machines. It is important that this surface man have adequate assistance for his work as well as comfortable and well lighted quarters with hot and cold water within easy reach, for his work is no less important than that of the man who is in charge of the actual rescue operations underground.

3. It was proven to be very desirable to have an intelligent man, who is thoroughly acquainted with the mine and the surface thereof, whose duty it is to receive orders from the rescue force and promptly procure and deliver the supplies desired. Without the proper man for this service great confusion results from the misunderstanding, or perhaps forgetting of the orders of the underground force, the delivery of insufficient amounts of materials and the delivery of materials to the wrong places. This man should insist that all work, such as the sawing of timbers to proper lengths, or the thawing of clay, be done on the surface in order to expedite the underground operations.

4. Two makes of rescue apparatus, the Draeger and Fleuss Proto, were in use at this fire. Two models of the Draeger were used, the 1910 and the 1914, the latter being equipped with the by-pass valve and mouth breathing attachments. Both the Draeger 1914 model and the Fleuss gave excellent results, but, as might be expected, in some particulars one model excelled while in other particulars the advantage was with the other.

The old 1910 type Draeger machines gave good service during the early stages of the fire when neither of the other types were available, except in two particulars, that of the helmet or face mask, the air cushions of which were subject to leaks, and the inability to secure an additional supply of air quickly due to the absence of a by-pass valve. This machine should undoubtedly be considered an obsolete type.

The 1914 model Draeger gave almost perfect service. Five of these machines were in continuous use for a period of 72 hours without a single machine being taken to the surface for repairs, adjustments or recharging. These machines were equipped with both helmets and the mouth breathing attachments, the preference of the men being largely in favor of the mouth breathing attachment. This machine had the advantage over the Fleuss machine in that it could be recharged underground thus making it possible to keep them in continuous operation. Another advantage noted in the use of this machine was at the time when a temporary bulkhead on the 41st level blew out due to the pressure of the heated gas and smoke behind it; here it was necessary for the men to replace the bulkhead under the most extreme conditions of heat and gases, the heat being so great that they could continue the work for no longer than five minutes at a time. Working under these conditions the respiratory air became insufferably hot due to the rapid chemical change in the regenerating cartridge. This condition was remedied by inserting at frequent intervals a fresh and cool regenerating cartridge. The cost of the cans so used may appeal to some as being extravagant, yet it was after all insignificant when compared to the property at stake. The same heating of the regenerating material would, of course, occur in the Fleuss machine under similar conditions but due to length of time and the difficulty involved in recharging the regenerating material it would have been necessary under similar conditions to have breathed overheated air, or perhaps to have had a great number of machines in operation.

5. The Fleuss machine demonstrated that it has a very definite advantage in the cost of operation. Furthermore it was proven that in case the standard brand of caustic soda is exhausted that the ordinary commercial grade, which in most places can be procured locally, may be used with complete success. It may as well be said of the Fleuss machine that it was less liable to disorder than the Draeger.

One disadvantage noted in the operation of the Fleuss machine was in the matter of the "freezing" or setting of the caustic soda in the bag after use. At times the men engaged in the actual apparatus work were on duty 15 minutes with a 30 minute rest between working periods. During this rest period the caustic soda would in some cases "freeze" so solidly that it was thought dangerous to re-use the machine, which involved sending the machine to the surface where the bag could be freed of the mass by the use of hot water.

6. One small item, but perhaps worthy of note, was the persistent use of dust respirators by shaft men at the time of the fire; these are of no value in a gaseous atmosphere, yet to the untrained man they give a false sense of security and consequently in several instances men using them ventured too far which resulted in their incapacity for at least the remainder of the shift. They should be barred from the danger zone at all mine fires.

7. As to the character of the gases in which the helmet men were compelled to work, this may be best judged by the statement that at one point where the gases were not thought to be especially bad, a canary bird died in three minutes.

8. The magnitude of the operations can best be judged by the number of machines in actual use; at one time there were 25 in use with five in reserve on surface.

9. The rescue work was carried on in places where the common miners' carbide lamp would not burn. The common two and three cell Ever-Ready flash light was used with complete success and no rescue equipment should be considered at all complete without an adequate supply of these electric lights. Two large "King" fire chief electric lights were used and gave excellent satisfaction while their charge lasted; they were especially good where the smoke and gases filled the air but they have the disadvantage of being difficult to recharge.

10. The fire proved the wisdom of having in and about the mines a number of men who have been previously trained in the use of rescue apparatus. The men so trained by the Calumet & Hecla gave splendid service and their skill in the use of these machines made it possible to control the fire without serious loss of property which undoubtedly, but for them, would have reached the shaft and in this event the loss would of course have been serious.

BLASTING EXPLOSIVES AND THEIR ACCESSORIES

BY CHARLES S. HURTER, WILMINGTON, DELAWARE.*

In compiling this paper the writer has endeavored to give a clear understanding of the nature of blasting explosives and their accessories. While technicalities have been avoided, as far as possible, it is only intended for those who have the knowledge of chemistry that goes with an engineer's education and for those who have had practical experience in the use of explosives. It is based on a study of the principal technical works and treatises on explosives including Berthelot, Guttman, de Kalb, Walke, the reports of His Majesty's Inspectors of Explosives (English), the publications of the United States Bureau of Mines and on the writer's own investigations and practical experience. An attempt has been made to select from all these sources all that is pertinent to our subject in its applicability to the explosives in general use. It has not been found practical to give each authority full credit for the material taken from his works, but the writer gladly acknowledges his deep indebtedness to all of them. However, he in particular wishes to express his appreciation of the kindness of Mr. R. L. Oliver, of the California Cap Company, who reviewed the entire paper and wrote the section on detonators.

Explosives are classified in several ways, that is either according to the effect they produce, the purpose for which used, or to their constitution.

The customary way is to speak of them as "High Explosives" and "Low Explosives." Those in which the chemical transformation is so very rapid that its effect, when they are detonated by blasting caps, is a violent local and shattering one, are considered High Explosives; those in which the transmission of the chemical action proceeds comparatively slowly

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through the mass, when simply ignited by a spark, and the effect produced is a heaving or propelling one, are classified as Low Explosives.

No hard and fast line can be drawn between the two; dynamites are characterized as High Explosives, while Black Powder is a type of Low Explosives. Practically all sporting and ammunition powders belong to the low explosive class. Heavy ordnance frequently contains both classes: low explosives being always used to propel the projectile out of the cannon, but the projectile itself, if it is to contain a bursting charge, is loaded either with a high or low explosive as required. For instance, if a disruptive effect is desired, it contains a high explosive charge; if on the other hand, a driving or spreading effect is wanted, as in the case of shrapnel, a low explosive is employed.

Fulminate of Mercury, and mixtures, such as are commonly used in detonators, in which this substance is the base, some authorities classify in a separate group called "Fulminating Explosives," implying that they are extremely sensitive to friction, percussion or simple ignition and have a violent local effect. Their action being an extreme instance of rapid chemical transformation they are consequently a very high explosive. Nitrogen Iodide, Silver Oxallate, Nitrogen Sulphide and others, too sensitive and dangerous to consider, are also included in the category of fulminating explosives.

The constitution of explosives is sometimes used for arriving at a basis of classification, as in the case of mechanical compounds. Low explosives are thus classified as "Mechanical Mixtures," of which black powder may be taken as an example. It comprises combustible bodies and a supporter of combustion brought into very close contact by means of mechanical mixing. The explosive action takes place by rapid process of oxidation, or combustion set in motion by means of a hot spark or flame. The particles composing the different substances in the mixture react upon each other and propogate their explosion from one group of particles to the next and so on throughout the entire mass. An explosion so prduced is necessarily relatively slow.

As a contrast to this, the action of high explosives when detonated by a blasting cap or other medium communicating an initial shock, consists principally of a violent dissassociation of chemical compounds followed by an intense oxidizing action. The elements that make up the principal commercial high explosive are all in definite chemical combination constituting definite molecules of uniform composition held together by a relatively feeble attraction. A shock will overcome the bonds which hold these elements together thus freeing them instantly to react upon each other, and so to produce large volume of gases generating great heat. This reaction takes place with such rapidity and energy that it creates a strong explosive effect. The reaction thus started will be continued throughout the entire mass, and if the initial shock be provided through the medium of a suitable detonator calculated to effect complete reaction in an infinitesimal fraction of time, an effect of particular violence is obtained, which is called "Detonation."

This subject of detonation is a broad one and will be discussed later on in this article in connection with priming charges and conditions suitable to obtain the most effective results with the various types of explosives in commercial use. It is deemed expedient to study the characteristics of the various explosives first.

BLACK BLASTING POWDER.

"A" Blasting Powder (Nitre) is made from Potassium Nitrate, Charcoal and Sulphur, in the approximate proportions of 75, 15 and 10. It is used mainly for blasting hard dimension stone and for work in damp climates.

"B" Blasting Powder (Soda) is made from Nitrate of Soda, Charcoal and Sulphur in the approximate proportions of 72, 16 and 12. Because of its lower cost, "B" powder is more commonly used than "A" powder, and "B" is sufficiently strong for most of the purposes for which blasting powder is used.

Theoretically, "B" Blasting Powder should be a little stronger than "A" Blasting Powder on account of the higher oxygen content of Nitrate of Soda, (Nitrate of Soda, 56.4% Oxygen; Nitrate of Potash 47.5% Oxygen) which allows the use of a greater percentage of combustible material, but by actual tests the "A" powders shoot a little stronger. This is the opposite from the dynamites, where the insoluble ingredients form sufficient protection against the action of moisture to cause the "Soda Dope" explosives to shoot stronger than the more expensive varieties containing Nitrate of Potash.

The formula of Black Blasting Powder does not provide for complete combustion as it has been found that, when it is made up to accomplish this, the powder is not so strong because the increased temperature does not make up for the smaller amount of gas formed than is the case when the gases contain both carbon dioxide (CO_2) and Carbon mon-oxide (CO) instead of Carbon dioxide alone. The formula used is the result of practical tests to obtain the greatest efficiency and not from calculations from a standpoint of chemical reactions.

Important Properties—Black Blasting powder is not made in different strengths like dynamite, but varies in quickness depending upon the size of the grains. Classes A and B are furnished in different granulations. "A" Blasting is made in the following granulations: C, F, FF, FFF, FFFF, FFFFF, FFFFFF, and FFFFFFF, but the sizes most commonly used are C, F, FF and FFF. "B" Blasting Powder sizes are CCC, CC, C, F, FF, FFF, and FFFF. The CCC grains which represent the largest size, are about one inch and a half in diameter, and the FFFF grains the smallest, are about 1/16 inch in diameter. The finer granulations are much quicker than the coarse and are used for blasting rock, refractory materials and coking coals. The coarser granulations are slow and are used for other coals, earth work and shales or wherever it is desirable to heave out material in large pieces instead of shattering it. Most manufacturers make a special granulation of "B" Blasting Powder for large blasts called RR. This consists of a mixture of FF, FFF and FFFF grains designed to fill the spaces between the large grains with small ones and thus obtain a compact concentrated load. The load is about 10% denser with the RR granulation than with any single size.

Practically all the Blasting Powder sold in this country is glazed (polished). This is a coating of graphite which offers a little resistance to moisture and makes the powder free running, thus giving a better load, but aside from this it adds nothing to the efficiency of the powder.

The specific gravity of Black Blasting Powder varies from 1.5 to 1.9, usually being about 1.8. High specific gravity results from compressing the powder to smaller bulk with consequent reduction of air spaces of the individual grain. Black Powder is unaffected by cold, but has little resistance to water, since the nitre is readily soluble.

NITROGLYCERIN.

(Berthelot and Willis A. Hill).

The chemical formula for nitroglycerin is $C_3H_5\ (NO_3)_3$. Its specific gravity at 60° F. is 1.599. Its weight per quart at 60° F. is 3 pounds 5 1/3 ounces. The freezing point is 46.4° F. It is very soluble in alcohol or ether but only slightly soluble in water. It is poisonous. The action of solar light causes the decomposition of nitroglycerin, as well as that of nitric compounds in general. This is the reason why dynamite should never be thawed by placing it in the warm sunlight.

Submittted to the action of heat, nitroglycerin is volatilized to an appreciable extent, especially toward 212° F; it may even be completely distilled, if this temperature be long maintained, but if the temperature be suddenly raised to about 392° F., nitroglycerin ignites and a little above it explodes with great violence. Also, perfectly pure nitroglycerin will not stand a temperature of 212° F. for more than a few hours without decomposition and possibly explosion.

Its inflammation, caused by contact with an ignited body, gives rise to nitrous vapor and a complex reaction, with the production of a yellow flame without explosion, properly so-called, at least as long as small quantities of matter are operated upon, but if the mass be too great, it burns in layers until the lower body arrives at a temperature of 392° F., which is the exploding point.

One kilogram of Nitroglycerin should give 1135 litres of gaseous products. The temperature of detonation of nitroglycerin is 3005° C, or 5441° F. The quantity of energy given off by 1 kilogram is 6050.48 kilogrammeters; by one pound 19850.58 foot pounds. The per cent. by weight of products of detonation are by analysis:

Carbonic acid	CO_2	57.6
Oxygen	O_2	2.7
Nitrogen	N_2	18.8
Water	H_2O	20.7
		99.8

The specific heat of liquid nitroglycerin is 0.4248.

The latent heat of frozen nitroglycerin (latent heat of fusion) is 33.5 calories.

Thoroughly washed nitroglycerin does not lose its strength with age.

On freezing, nitroglycerin expands 10/121 of its entire volume which is greater than any other liquid.

DYNAMITE.

The different types of high explosives used for blasting vary widely in their properties. Some are exceedingly quick, some relatively slow, and others are intermediate in quickness. Different types of explosives also vary in density from the heavy gelatins to some of the coal mine powders, which are very light. High explosives are graded according to their strengths compared on a weight for weight basis with the straight nitroglycerin dynamites, which are the only type containing the actual percentages of nitroglycerin as designated. The other types make up their strengths by the use of such explosive substances as organic nitrosubstitution compounds, explosive salts and guncotton.

The high explosives used for blasting in the United States are divided into three popular classes: Straight Dynamite, Ammonia or so-called "Extra" Dynamite, and Gelatin.

The First Dynamite that appeared on the market was composed of 75% nitroglycerin and 25% of an infusorial earth called kieselguhr. The kieselguhr being an inactive substance, consumes some of the strength of the nitroglycerin and detracts from its sensitiveness. The result is that the 75% kieselguhr dynamite is only about as strong as the American 40% dynamite, and kieselguhr dynamite containing less than 40% of nitroglycerin cannot be exploded at all by ordinary means.

Another of the earliest absorbants used for nitroglycerin was sawdust. It was from this that the American Dynamite of the present day was evolved. Theoretically, when nitroglycerin is detonated, 2.7% of the gases liberated is free oxygen, which enabled the sawdust absorbant dynamites to show slightly greater strengths than the corresponding kieselguhr dynamites. The present American straight dynamites are made up with a dope of nitrate of soda and wood meal, having proportions designed to bring about complete combustion through a complete balance of the chemical reactions which take place on detonation.

The Straight Dynamites containing only nitroglycerin, nitrate of soda, wood meal and antacid are taken as standards because they are the oldest, simplest and best known type of high explosives in the United States. They are more or less pulpy and easily crumbled when the wrappers are removed. Straight dynamite can be manufactured in strength from 12½ to 75%. However, 15% is the lowest practical limit and strengths above 60% are banned by the Bureau for the Safe Transportation of Explosives Rule prohibiting the acceptance by railroads of explosives containing more than 60% of nitroglycerin in a liquid form as it is in the straight dynamites.

Straight dynamites are suitable for use requiring strength and quickness when the water conditions are not too severe. The fumes from them are the worst of the three classes of dynamite. The weaker grades, however, give off less deleterious gases than the strong. The straight dynamites are very easily ignited by flame or sparks such as might issue from the sides of defective or the cheaper grades of fuse. Therefore, considerable care must be taken in the making and placing of primers when using straight dynamites for blasting.

Ammonia Dynamites—By replacing approximately half of the nitroglycerin of the straight dynamites with nitrate of ammonia, another series of explosives has been created having the same strengths as the straight dynamites, but possessing certain distinguishing characteristics that make them very valuable for certain classes of work.

The ammonia dynamites have a somewhat slower action because they contain less nitroglycerin than the corresponding grades of straight dynamite. The fumes are much less obnoxious than those of the straight dynamites and they are practically as good as those from the gelatins which will be described later. The ammonia dynamites are the most difficult of any of the ordinary high explosives to ignite. It is practically impossible to ignite ammonia dynamite from the side spitting of fuse. A strong blasting cap must be used as a detonator.

The solubility of nitrate of ammonia affects the water resisting qualities of this type of dynamite but they are brought back to a resistance almost equal to the sraight dynamites by coating the nitrate of ammonia with petrolatum and dipping the cartridges, after filling, in melted paraffin to seal them from the effects of moisture.

Gelatin Dynamites—Alfred Nogel discovered in the course of his research work with explosive compounds that some of the lower grades of guncotton could be dissolved in nitroglycerin, forming a waterproof jelly that is slightly stronger than nitroglycerin. This jelly is the base of the present gelatins or gelatin dynamites. It also forms an impervious protection to the soluble ingredients against water.

The gelatins are distinguished by their plasticity, high density, imperviousness to water and comparative freedom of their explosive products from obnoxious fumes. When detonated by means of a blasting cap, their action is much slower than the straight or ammonia dynamites, but this deficiency is largely overcome by the concentrated load due to their density and plasticity. On the other hand, a quickness about 10% stronger than the corresponding grades of straight dynamite may be effected by detonating the gelatins with a primer of very strong nitroglycerin dynamite or blasting gelatin.

The gelatins are the densest of the high explosives which fact makes them very valuable for tight blasting where a concentrated charge is desired at the bottom of the holes. The fumes are superior to those from both the straight and ammonia dynamites. Being very nearly water proof in the lower and perfectly waterproof in the higher grades, the gelatins are best adapted for use in extremely wet work and submarine blasting.

Granular dynamites form a class that can be considered as the connecting link between blasting powder and dynamite. They are known as Judson Powders and Low Powders. This class consists of a series of explosives containing small percentages of nitroglycerin with a dope designed to allow the nitroglycerin to exert its full explosive effect. The dope consists of hard impervious grains composed of sulphur, nitrate of soda and high grade blacksmith coal melted together and broken up to a size about the same as FFFF blasting powder. The nitroglycerin is present as a coating on the grains and is held between them. The lowest grade known as RRP contains 5% of the nitroglycerin while 20% represents the maximum amount of nitroglycerin than can be held between the grains of the dope. The RRP should have a primer of 40% or stronger dynamite in proportion of at least one pound of dynamite to 25 pounds of RRP. Ten per cent. and higher strengths can be fired by means of blasting caps.

The RRP is generally packed in 12½ pound bags, four bags to the case. It is free running and can be easily poured into sprung holes, etc.

These granular dynamites are valuable for blasting in open work that is comparatively dry and where black blasting powder is not quick or strong enough. They are not recommended for underground work on account of their fumes. They are very inflammable and when shot by cap and fuse care should be taken not to allow the fuse to come in contact with the powder. The 10% and higher grades are packed in cartridges the same as dynamite.

Permissible Explosives.

The subject of Permissible Explosives and their tests has been so thoroughly covered in the publications of the United States Bureau of Mines that the writer does not consider it necessary to take up this subject in detail. The two principal classes, as far as practical results are concerned, are the nitroglycerin and the nitrate of ammonia permissibles.

There is a limit to the strength of permissible explosives. The regular dynamites have two sources of strength, the volume of gas given off and the temperature of explosion which adds to their expansive force. In the permissible explosives the formulae are designed to keep the explosion temperature as low as possible and consequently no permissible explosive has ever been manufactured that will do the work of high grade dynamite.

The nitroglycerin class is simply nitroglycerin dynamites containing an excess of wood meal or other carbonaceous matter. In them the object is to burn the carbon to carbonic oxide (CO) instead of carbon dioxide (CO_2). In short, their temperature of detonation and amount of flame is reduced by applying the principle of incomplete combustion. These explosives can be graded in different strengths below a 35% dynamite. They are the best for work too wet to allow the use of the ammonium nitrate class.

The ammonium nitrate permissible explosives, according to their name, have ammonium nitrate as their principal ingredient. Ammonium nitrate, as an explosive ingredient, has the advantage of being composed of entirely gaseous ingredients, namely, Nitrogen, Hydrogen and Oxygen. and consequently there is no loss due to the formation of a solid residue as is the case with sodium nitrate. Ammonium nitrate, in an explosive, has a very small oxidizing action. This can be expressed as follows:

$$\underset{}{NH_4\,NO_3} = \underset{\text{Gaseous}}{2H_2O} + \underset{\text{Gas}}{N_2} + \underset{\text{Gas}}{O}$$

In comparison with this the explosive reaction of nitrate of soda is as follows:

$$2Na\,NO_3 = \underset{\text{Solid}}{Na_2O} + \underset{\text{Gas}}{N_2} + \underset{\text{Gas}}{5O}$$

In other words nitrate of ammonia as an explosive has only 40% of the oxidizing power of nitrate of soda, consequently the carbonaceous matter required is 60% less which results in the smallest possible heat producing reaction. Also a large amount of heat is absorbed in the production of the water in the gaseous form. Therefore, the principles involved in the manufacture of ammonium nitrate permissibles are the small but complete combustion and the latent heat of steam. In this manner the presence of deleterious gases after explosion is avoided.

Ammonium nitrate permissibles being bulky, a shipping case averages about one-third more cartridges of a given size than is contained in the same weights of dynamite. A 50-pound case of dynamite averages about 100 cartridges 1¼x8, ammonium nitrate permissibles 135. The result is that while weight for weight, ammonium nitrate permissibles are equal in strength to a 60% dynamite, cartridge for cartridge, they correspond to a 30% to 35% dynamite. The quickest varieties are the strongest.

The most successful ammonium nitrate permissibles contain about 10% of nitroglycerin as a sensitizer. With the exception of the slowest varieties they can be used when frozen, provided the priming cartridge is loosened up enough by rolling, to permit the proper placing of the detonator.

The fact that each cartridge of the quickest ammonium nitrate permissibles is equal to the same sized cartridge of 35% dynamite has been taken advantage of by a large number of users of explosives other than coal miners. They are being used in a large number of metal mines, quarries and all kinds of open work, where the water conditions are not too severe, with more economical results than could be obtained with 30% to 40% dynamite.

There are a few permissible explosives which do not belong to the above mentioned classes. Most of these use salts containing large percentages of water of crystallization. The vaporization of this water while it reduces the flame and explosion temperature also cuts down the strength of the explosive and reduces its sensitiveness. One example of this was an explosive on sale 10 or 15 years ago, containing 40% of nitroglycerin and so much alum that its strength was only equal to a 15% dynamite. This type of explosive is only manufactured where the demand is sufficient, to meet special conditions where a weak explosive can be used.

Low Freezing Dynamite.

Ordinary nitroglycerin dynamite freezes at temperatures between 45° and 50° Fahrenheit. There are a number of nitrosubstitution organic compounds, such as nitrotoluene, nitrobenzol, etc., that, when dissolved in nitroglycerin, have the effect of lowering its freezing point in the same manner that salt lowers the freezing point of water. The first low freezing dynamites, and similar explosives, made use of this principle exclusively. They would not freeze at temperatures above 32° Fahrenheit. Since that time the freezing temperature has been further reduced by some secret manufacturing processes that the manufacturers are unwilling to divulge. The first low freezing explosives had a peculiar aromatic odor that was very noticeable, but during the last few years this has been avoided.

The present low freezing explosives will not freeze until after water freezes. In addition to being low freezing they are also slow freezing, that is, after their freezing point has been reached, it may take anywhere from several days to a month for them to become hard.

No definite freezing temperature can be given for these explosives. It is variable over which the manufacturer has no control. Sometimes they will remain soft for a long time at a temperature of zero Fahrenheit, while another lot will congeal at a higher temperature. All that can be said definitely is that low freezing explosives will not freeze until after water freezes.

DETONATORS.

BY R. L. OLIVER.

In practice, the detonation of high explosives is accomplished by means of an intermediate agent that will produce a violent impulse embodying both shock and heat.

Fulminate of mercury forms the basis of most of the commercial detonator charges. According to Berthelot, the impulse from fulminate of mercury is quicker and more intense for a given volume than that of any other substance used for producing detonation. This is explained by the suddenness of its decomposition, together with the extraordinary magnitude of the pressure which it would develop when detonating in its own volume. This is given as 2,600 atmospheres or 38,220 pounds per square inch. Other fulminating explosives in conjunction with fulminate of mercury have been employed in commercial detonators during the past decade with very satisfactory results, and these will be discussed in another part of this paper.

The action which takes place between a fulminating detonator and the blasting charge to be detonated depends more or less on the strength of the initial pressures, on the suddenness of their development and the relative stability of the compounds used to make up the explosive, which, in turn, regulates the ease by which the shock is communicated to the rest of the mass. That is to say, the action which takes place depends on the conditions which regulate the energy transformed into heat in a given time on the first layers of the explosive substance reached by the detonator.

The quantity of energy thus transformed depends, therefore, both on the quickness of the shock and the amount of work it is capable of performing. This gives us two conditions which vary with each explosive substance to be detonated. Thus the most suitable priming mixtures and compounds are not always those in which the explosion is quickest. For instance, nitrogen chloride and nitrogen iodide are not as strong primers as fulminate of mercury although they are much quicker in their explosive action.

The chemical reaction that takes place on the detonation of fulminate of mercury is expressed as follows:

$$Hg\ C_2\ N_2\ O_2 = 2\ CO + N_2 + Hg$$

According to this equation, only carbon monoxide, nitrogen and mercury vapor are formed. One only of these is a compound; it is stable and not susceptible of dissociation. Moreover, the total heat of decomposition is disengaged at once and the gases are produced without the occurrence, during cooling, of any progressive recombination that would tend to moderate their expansion and diminish the violence of the first shock. The condensation of mercury vapor can be disregarded as this only takes place at temperatures below 680° Fahrenheit.

Mixtures of fulminate of mercury with other compounds are made with the object of increasing its sensitiveness to ignition and also for increasing its expansion to extend its effectiveness; detonating in its own volume, as it does, the effect of straight fulminate is too local. The compound most commonly used for the above purpose is chlorate of potash. The decomposition of chlorate of potash into potassium chloride and oxygen liberates heat and the conversion of carbon monoxide into carbon dioxide also generates heat, the effect of which increases the pressure of the gases formed and also intensifies the chemical reaction on the explosive to be detonated. This oxidation being a secondary reaction retards the velocity of the fulminate somewhat, but heat as well as suddenness of shock being a contributing element of detonation, it has been shown to be advantageous to use mixtures of fulminate with other ingredients where the great gain in heat and expansive qualities more than counterbalance the rendering of the fulminate itself any less sudden in its action, the resultant effect being an intensely violent explosive impulse or wave.

The chemical reaction involving fulminate of mercury and chlorate of potash for the complete combustion of their products, is expressed according to the following equation:

$$\underset{852.12}{3\,Hg\,C_2\,N_2\,O_2} + \underset{245.2}{2\,KCl\,O_3} = 6\,CO_2 + 3Hg + 2KCl + 6N$$

At the detonation temperatures of fulminate of mercury the oxides of mercury cannot exist.

According to this reaction 852.12 parts of fulminate of mercury are mixed with 245.2 parts by weight of chlorate of potash to bring about complete combustion. This proportion can be expressed as 78% fulminate and 22% of chlorate. The mixtures in use have varied from 90% of fulminate of mercury with 10% chlorate of potash to 80% fulminate with 20% chlorate.

The Bureau of Mines, after exhaustive tests, recently showed that the 80/20 mixture created a more violent impulse than either the 90/10 mixture, or even straight fulminate, which confirms the writer's foregoing explanation of the elements contributing to an effective initial detonating impulse.

It has also been shown beyond question of doubt, by the Bureau of Mines and other investigators, that certain nitrosubstitution compounds such as nitrovene, trinitrotoluol, tetranitromethylanalin, tetranitroanalin, and nitromannite, each with a small quantity of fulminate-chlorate mixture as a primer in a reinforced capsule, make more efficient detonators than fulminate and chlorate alone for modern dynamites that contain mixtures of nitrosubstitution compounds, with or without nitroglycerin, such as the ammonia dynamites, gelatins, low freezing and permissible explosives.

These dynamites containing nitrosubstitution compounds are less sensitive, hence harder to detonate than straight nitroglycerin dynamites. The fulminating nitrosubstitution detonators, being of lighter specific gravity than the fulminate-chlorate mixtures explode with the liberation of more heat and in a larger volume, thus distributing their initial shock over larger area of the dynamite primer. The detonating wave thereby created, although applied less locally, acts more violently upon the mass to be detonated, and inasmuch as the detonator that produces the best results is what counts most in practice, regardless of what it contains, these composition detonators are being received with favor.

Fuse.

The English Explosives Act of 1875 contains the following:

"The term 'Safety Fuse' means a fuse for blasting which burns and does not explode, and which does not contain its own means of ignition and which is of such strength and construction and contains an explosive of such quantity that the burning of such fuse will not communicate laterally with other like fuses."

The body of the fuse, or base upon which all grades are built, with few exceptions, consists of ten strands of jute thread wound round a core of fine grained potash black powder. The powder is fed through a funnel at the same rate that the jute threads are spun into the cord. In order to prevent the powder from clogging the point of the funnel, a fine cotton thread, known as the core thread, is fed through the funnel and into the core of the fuse. This core thread can be found in the powder train, of all fuses, but it has absoultely no effect on the burning speed of the fuse.

To the jute centre the first layer of waterproofing material is added, on this base the fuse is built up as desired. The waterproofing compounds used are tar, asphalt and gutta percha. These substances are all very efficient; their drawbacks are that tar and asphalt harden and become brittle in cold weather. Gutta percha, unless protected from direct contact with the air, oxidizes rapidly.

From the jute centre the fuse may be built up by the use of tape windings, making single, double or triple tape fuses as the case may be. The tape fuses, as a rule, have tar or asphalt for waterproofing. The jute centre again may be wound with fine cords impregnated with gutta percha. The cord winding is known as countering and when countered fuse is referred to it means a fuse that is made up of cord wound layers.

The principal differences between the tape and countered fuses are that the tape fuses will stand more abrasion and scraping, but are very liable to break when bent sharply and in cold weather they become hard and brittle. The countered fuses are more pliable and do not beccome hard in cold weather.

Crimping Blasting Caps on Fuse.

This operation should be done only by means of an approved tool made especially for this purpose and known as a cap crimper. The fuse should be cut square and placed in the cap so that it barely touches the charge. Then fasten the cap firmly on the fuse by pinching it close to the open end with the crimper. With the broad type of crimper such as is recommended by most manufacturers, the crimp should include the edge of the open end of the cap.

This crimp is not air tight. After a large amount of experimental work, several years ago, the leading powder companies produced a crimper that made an absolutely air and water tight crimp. This was followed by a number of complaints of misfires which at first could not be explained.

One of the principal fuse manufacturers made the following experiment: A piece of dentist's sheet rubber was fastened air-tight on one end of a 3 ft. length of fuse. Shortly after the other end of the fuse was ignited the rubber began to swell, and before the fire had gotten within eight or nine inches of this end of the fuse, the pressure was sufficient to burst the rubber.

One of the powder companies followed this up with another experiment. Some special blasting caps were made up with one grain of fulminate loaded in a No. 8 shell (2⅛ inches in length). Some fuse previously tested to show a strong end spit was also used. An air-tight crimper and a broad type were used. The fuses were crimped at varying distances from the charge and as soon as the fuse, which was fastened with the air-tight crimp, was moved back to a position not in close contact with the charge, misfires began to occur. With the broad type, while it is recommended that the end of the fuse be close to the cap charge, no misfires occurred, the flash extending the entire length of the capsule when the fuse had only enough length in the shell to permit crimping. To sum up, these experiments showed conculsively that when fuse burns some of the gases proceed ahead of the fire through the jute spinning threads which form the inner body of the fuses. Also that when an air-tight crimp is made these gases may develop a pressure inside the blasting cap that will not only prevent the flash or end spit of the fuse from coming out into the cap, but force it back through the fuse. Therefore, a vent must be provided for the escape of these gases and thus allow the full end flash of the fuse to blow into the blasting cap and explode it.

In wet work some protection must be provided to prevent the entrance of water until the gas pressure is sufficient for this purpose. The fuse companies manufacture a compound for this purpose. Some use P. B. paint by dipping the caps, after crimping, in the paint and allowing them to dry. It has been found that hard cup grease, such as Albany No. 3, paraffin wax, (Sunshine) and tallow are good for this purpose, but on the other hand soft grease, engine oil, cylinder oil, mineral grease, vaseline, etc., are liable to strike through the fuse and injure the powder train if left standing.

There is also on the market a special cap protector, that consists of a length of pure rubber tubing of the proper diameter to make a tight fit over the joint between the cap and fuse. These come rolled on a stick in such a manner that they can be easily transferred to a blasting cap and straightened out after the fuse has been inserted and the cap crimped in place.

Sensitiveness to Detonation.

The sensitiveness of explosive substance to detonation depends upon the relative stability of the ingredients, the temperature of the explosive and the modes of propagation of the explosive reactions. Thus, for example, silver oxallate detonates at about 266° Fahrenheit, nitrogen sulphide at about 405°, and fulminate of mercury at about 374°. Nevertheless, fulminate of mercury is more sensitive to friction than either nitrogen sulphide or oxallate of silver. Therefore, it can be said that special properties, depending on the chemical structure of each substance, particularly in solids, favor decomposition under given circumstances.

There are also some general conditions which affect the sensitiveness of commercial high explosives to detonation. The sensitiveness of any explosive substance increases with the initial temperature at which the explosive reaction begins; or in other words, its sensitiveness to detonation becomes greater as the temperature is approached at which the body commences to decompose spontaneously. The explanation of this is that the heat liberated by the explosive reaction proper undergoes less loss by radiation, therefore, a greater weight of the non-decomposed substance is raised to the desired temperature at the beginning of the explosive reaction. These facts, especially that in regard to the temperature of the explosives, can be considered as some of the primary causes of accidents in the thawing of nitroglycerin explosives, when they are allowed to become overheated by carelessness or the use of improper methods of thawing.

The sensitiveness to detonation of an explosive substance will be rendered still greater if this temperature limit is exceeded; that is, if conditions prevail under which a slow decomposition may be transformed by the least shock or additional heating, into a rapid decomposition. A substance taken at a point near or above this limit may be considered to be in a state of chemical tension.

Sensitiveness to detonation also depends on the quantity of heat liberated by decomposition. That is to say, that all other things being equal, the explosive substance that liberates the most heat is most sensitive to detonation. To go into detail one can easily conceive that if, with an explosive that generates a large amount of heat, a small portion is brought to the temperature of detonation, it will communicate the explosive reaction throughout the entire mass much quicker and more completely than would be the case of an explosive that liberates a small amount of heat.

The same quantity of heat will produce different effects on the same weight of matter, according to the heat conductivity of this matter. For instance, chlorate of potash, whose specific heat is 0.202, is a better conductor of heat than nitrate of potash whose specific heat is 0.239. Thus chlorate powders should be and are more sensitive than nitrate powders. This together with the lower temperature of decomposition of potassium chlorate and the fact that chlorate of potash by itself gives off oxygen and liberates heat at the same time, renders chlorate powders extremely dangerous, they being liable to spontaneous explosion at any time.

The structure of an explosive also has an effect on its sensitiveness. In the explosives containing nitroglycerin in a liquid form the principle of the incompressibility of liquids plays a very important part in their sensitiveness. The fact that liquids cannot be compressed especially under the quick severe shock of fulminate of mercury, the inertia of liquids causes them to act more like solids and in thus repelling the impulse of the fulminate, each succeeding particle of the explosive liquid becomes detonated. The larger the percentage of liquid nitroglycerin the closer the nitroglycerin particles will be together, the detonation will be propagated easier, and therefore the sensitiveness of the higher per cent. strength nitroglycerin dynamites is correspondingly greater.

On the other hand, blasting gelatin which is a rubber-like mass made by dissolving nitro-cotton in nitroglycerin, has a certain amount of give and can be compressed. In order to overcome this, a stronger detonator must be used to explode blasting gelatin and the other gelatins than is the case with the straight dynamites which contain liquid nitroglycerin. In fact, this deadening action due to the compressibility of the gelatin is such that it is impossible to make a gelatin of less than 35% strength that is sufficiently reliable to be detonated at all times.

The gelatins show a very interesting combination of explosive substances which increases the heat and consequently the strength of the respective explosives used in their manufacture. Roux and Serrau give the relative strengths as follows: Nitroglycerine 10; compressed guncotton 6.5. In the gases from detonated nitroglycerin there is, theoretically, 2.7% free oxygen. In the gases from detonated guncotton, there is 49.3% of carbon monoxide. (CO) By balancing the oxygen deficiency of the guncotton by means of the oxygen excess in the nitroglycerin, the extra heat units gained by producing complete combustion make the resulting blasting gelatin stronger than pure nitroglycerin.

In the ammonia dynamites, the sensitiveness is also lessened because of the reduction of the liquid nitroglycerin in contents in order to allow the use of the very stable nitrate of ammonia. Hence, nothing less than a No. 6 detonator should be used to explode either gelatin or ammonia dynamite. This rule is made compulsory by law in the European countries. The U. S. Bureau of Mines heartily recommends it.

The No. 6 detonator contains 1 gram (15.4 grains) of a mixture made up of 80% fulminate of mercury and 20% chlorate of potash, or of other ingredients which shall produce detonating qualities equal to the foregoing.

Compressed guncotton is less compact than nitroglycerin owing to its structure. The presence of spaces, however minute, causes the pressure due to shock to become sensibly attenuated. Guncotton as it contains these spaces is, therefore, more difficult to explode than nitroglycerin. Nitroglycerin is exploded by the fall of small weight from a given height onto an anvil; or by a very small weight of fulminate priming; or by the use of a primer charged with guncotton; whereas guncotton cannot be exploded by the drop of a weight, nor under the influence of nitroglycerin. It requires a very strong fulminate detonator to explode guncotton.

This principle of cushioning applies to the high explosives made up entirely of solid ingredients such as those with a base of either guncotton, trinitrotoluol, or nitrostarch. The spaces between the particles of these powders act as cushions and prevent the impulse of detonation from being communicated through them as easily as in the straight or ammonia or even the gelatin dynamites. Therefore nothing less than a No. 8 detonator should be used with this class of explosives which constitutes the bulk of non-freezing high explosives on the market at the present time.

Fifteen to 20% of water can be added to cellulose dynamite, rendering it insensible to the shock of a rifle bullet without depriving it of the property of being exploded by means of a strong blasting cap. When dynamite is in this condition nitroglycerin is exuded at the slightest pressure. Dynamite containing water is very greatly weakened as part of the heat of detonation, depending directly on the proportion of water mixed with the dynamite, is lost by the conversion of this water into vapor. This consumption of heat means a reduced expansion of gases and consequently a de crease in explosive strength.

Nitroglycerin, if it becomes ignited by some other means such as the side spitting of fuse just before the explosion of the fulminate detonator is less susceptible to the influence of the detonator. The advance burning of the nitroglycerin produces a void which prevents the fulminate from doing its work properly. The absence of immediate contact between a fulminate detonator and the dynamite in a priming cartridge is prejudicial against good detonation for the same reason, the shock being partly deadened by the interposed air.

The sensitiveness to the action of a detonator is greater in dynamite containing liquid nitroglycerin than in that containing frozen nitroglycerin. When nitroglycerin solidifies, like a large number of compounds, it crystallizes and thus tends to exclude all foreign matter in obeyance of the laws of crystallization. Accordingly, when dynamite freezes, the nitroglycerin tends to separate from the dope and, to this absence of homogeneity, Berthelot lays the reason for the insensitiveness of frozen dynamite. If dynamite is thawed slowly at a temperature not exceeding 80° to 85° Fahrenheit the nitroglycerin will be reabsorbed in the dope as perfectly as it was originally.

Detonation.

In the discussion of the detonation of explosives from a technical standpoint, the chemical reactions and physical phenomena that take place when explosives are completely and incompletely detonated, grade into one another in such a manner that it is hardly possible to treat these subjects entirely under separate headings.

Berthelot claims that there is no line of demarcation between "Explosions" of the blasting powder order and "Detonation" of high explosives, but that they represent the two limits of a wide range of explosive phenomena.

When an explosive substance is detonated in its own volume, the maximum of temperature and pressure, and consequently the maximum speed of the chemical reactions involved, is attained; that is, the total heat possible to be developed in the reaction is obtained at the instant that the energy of the explosion is exerted on the surrounding medium.

DeKalb says that in no case is detonation absolutely perfect under ordinary conditions, but this perfection is approached more closely according to the concentration of the explosive impulse due to good confinement.

Some experiments made by the Western Australian Government Commission and described in the English "Blue Book" of 1905, showed that the tamping of charges has a very marked effect on the proper detonation of the explosive used. When boreholes are tamped carelessly or when no tamping is used, the lack of confinement apparently causes a small part of the explosive to be detonated incompletely and consequently more offensive fumes are given off than when the charge is tamped properly.

Technical Paper No. 17 (The Effect of Steaming on the Efficiency of Explosives) of the United States Bureau of Mines describes experiments showing the gain in the work accomplished when dynamite is tamped. This varies from about 35% with the quick to over 90% with the slow acting explosives. The effect of tamping on fumes after blasting by the Western Australian Commission shows that the actual violence of the explosion is greater when the charge is tamped than when none is used.

As already mentioned, detonation is produced commonly in practice by means of a very sudden impulse, in which heat plays an important additional but secondary part. The gases formed at the point where the shock is first produced have not time to become displaced, so to speak, and in repelling the sudden blow, they immediately communicate their energy to the parts of the explosives in immediate contact. The action is thus propagated from particle to particle throughout the entire mass so intensely as to maintain in it a veritable explosive wave.

As a contrast to this, progressive combustion transmits itself step by step throughout the mass under conditions in which the cooling, due to conductivity by contact with the enclosing medium, lowers the temperature sometimes to the lowest degree compatible with the continuance of the reaction.

In regard to the effects of shock on nitroglycerin, it is sufficient to admit that the pressures, resulting from the shock administered on the surface of nitroglycerin, are too sudden in their action to distribute themselves uniformly throughout the entire mass. Consequently the transformation of energy into heat takes place, more especially in the first layers reached by the shock. If this shock be of sufficient violence, the first layers may be suddenly raised to a temperature about 392° Fahrenheit, which will cause them to be decomposed immediately, producing a great quantity of gases at a high temperature. This production of gas is so sudden that the adjacent particles of explosives have not time to be displaced and this sudden expansion of the gases produces a second shock, possibly more violent than the first on the layers of explosive situated below. The energy of this new shock is transformed into heat in the layers which it next reaches and thus causes their explosion.

This alternation between a shock developing an energy, which becomes changed into heat and which raises the temperature of the heated layers up to the degree of a new explosion capable of reproducing the shock and heat phenomena, transmits the reaction into the mass of the explosive a distance commensurate with "a" the intensity of the initial shock from the detonator and "b" the sensitiveness of the explosive to propagate its detonation thenceforth. Thus the propagation of the explosive action takes place by virtue of phenomena comparable to those which gives rise to a sound wave. The explosive wave is like the sound wave also in that the more intense the initial detonation produced the further it travels; and the greater the resistance, i. e. the more insensitive the explosive is, the sooner its detonation stops.

A true explosive wave is produced with a speed incomparably greater than that of simple inflammation effected under conditions which allow the gases to expand freely as they are produced.

The detonating wave travels with the greatest speed and to the greatest distance in *cartridge of large diameter* up to a certain limit. One and three-quarters inches appears to be the diameter beyond which the ease of detonation of dynamite is no longer sensibly increased. With cartridges of smaller diameter it is possible to decrease the size to a point where the explosive can no longer be propagated throughout the charge. As the ammonia dynamites and gelatins came into favor, supplanting the straight dynamites, it became necessary to abandon the manufacture of explosives having a cartridge diameter of less than 7/8 of one inch. There are quite a few of the dry ingredient high explosives that cannot be used in cartridges having diameters of less than 1 1/4 or 1 1/2 due to their relative insensitiveness.

The reaction induced by a given shock in an explosive substance is propagated with a rapidity which depends on the intensity of this shock, because the energy of the first shock transformed into heat determines the intensity of the first explosion, and consequently the intensity of the entire series of consecutive effects.

The intensity of the first shock may vary considerably according to the mode in which it is produced. It follows then that the explosion of a solid or liquid mass may be developed according to an infinte number of different reactions, each of which is determined by the original impulse. The more violent the initial impulse the more sudden will be the induced decomposition and the greater will be the pressure generated during the entire course of the entire decomposition. One single explosive may, therefore, give rise to the most diverse effects according to the method of its ignition.

These facts show the great importance of using suitable detonators in practice. In fact, Bichel strongly advises the use of a large excess of fulminate or other detonating ingredients at all times in order to overcome any bad effects due to possible moisture getting into the detonator, and also to the possibility of the detonator becoming loosened in the primer or being drawn away from it. Also enough extra fulminating mixture should be employed to insure the explosion of the dynamite cartridges in a blasting charge under adverse conditions of moisture, coldness, age, irregularities in tamping and air chambers between cartridges in charging bore holes, etc.

Methods of Preparing Dynamite Primers.

In regard to the making of dynamite primers, the accompanying sketch shows several methods that are in fairly

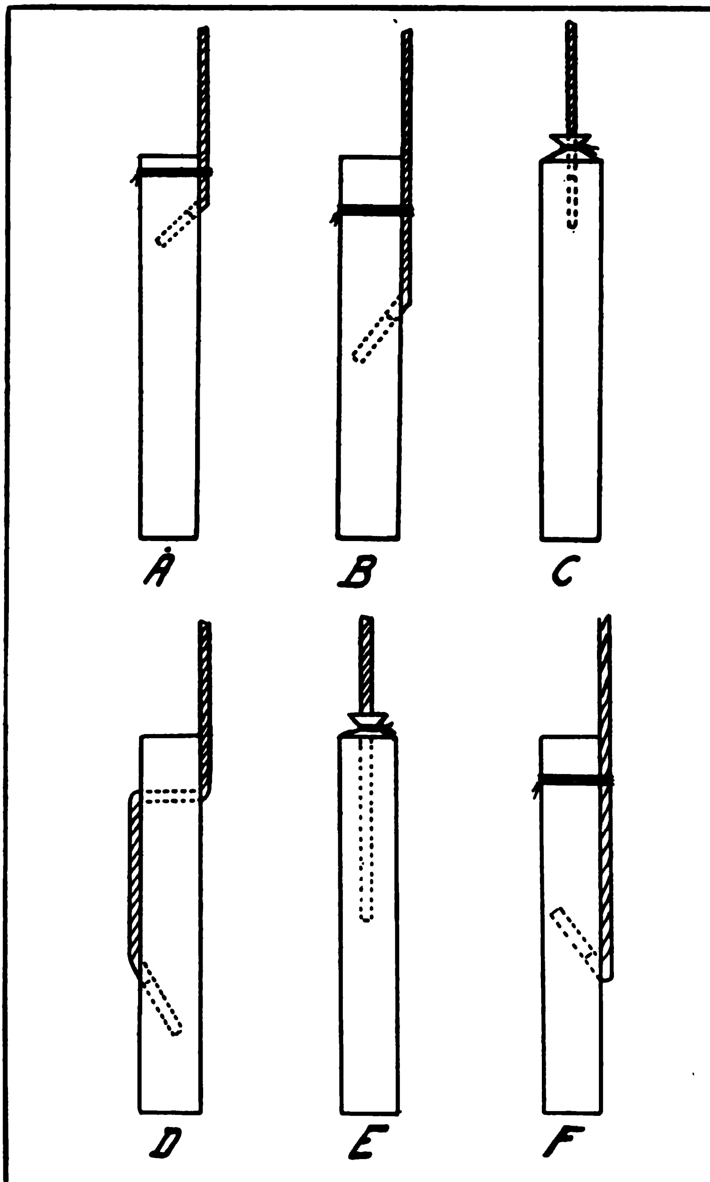


ILLUSTRATION OF METHODS OF FASTENING EXPLODERS TO THE DYNAMITE common use. Those marked A, B and C are considered to be good practice, as the blasting caps are not buried deeply in the powder and there is a minimum amount of fuse in actual contact with the explosive.

In methods A and B the blasting cap is inserted in the side of the cartridge and pointed diagonally downward; the fuse being tied firmly to the cartridge. These methods are about as safe as any for loading, as there is a small quantity of dynamite that acts as a cushion between the tamping stick and the blasting cap. Method B allows a little firmer tying of the fuse than method A.

Method C is the one recommended for wet work in particular and for holes where the cartridges fit tight enough to crowd out the fuse. This style of primer takes longer to make than the others but with care and the application of a little tallow or hard grease, this primer should be able to stand water from 15 minutes to half an hour and is especially recommended for wet work.

One of the greatest difficulties experienced in the manufacture of safety fuse is to make a fuse that does not spit fire from its side while burning. This is the cause of most of the burnt charges when the primers are made or placed so that considerable length of fuse is in contact with the explosive, as shown in the sketch at D, E and F.

The danger of trouble from the improper making of primer depends a great deal on the kind of explosive and the quality of fuse used. Methods A, B and C are considered safest and best for all conditions.

On the other hand the method of making primers as illustrated by D where the fuse is laced through the cartridges is no doubt the most common method in use.

Methods D and E are fairly safe with gelatins provided triple tape of a very high grade of gutta percha fuse is used. Methods D and E should never be used with straight dynamites, but with ammonia dynamites they are perfectly safe as it is impossible to ignite ammonia dynamite by the spit from any ordnary fuse.

Method F is to be avoided at all times as the sharp bend is liable to open the fuse so that water, if present, will cause a misfire. Also a sharp bend like this is liable to break the powder train and either cause the cap to misfire or the opened fuse might ignite the charge below the cap and form a gaseous cushion that would prevent the cap from detonating the explosive. Sharp bends in the fuse are always to be avoided in making primers or in the loading of holes.

In making primers great care should be taken that no part of the blasting cap projects from the outside of the cartridge. A large number of bad accidents have occurred due to the scraping of the blasting cap in an improperly made primer on the sides of the borehole.

The Western Australian Commission made a large number of experiments with straight dynamite. They state that a great deal of trouble in regard to powder burning and making bad fumes is caused simply by burying blasting caps and fuse too deeply in the priming cartridges.

The definition of safety fuse according to the English "Explosives Act of 1875" is as follows:

"The term 'Safety Fuse' means a fuse for blasting which burns and does not explode, and which does not contain its own means of ignition and which is of such strength and construction and contains an explosive of such quantity that the burning of such fuse will not communicate laterally with other like fuses."

In connection with this it may be easy to conceive that while a fuse may spit fire through its covering with an intensity that will not ignite a similar fuse placed beside it, this fire might easily ignite any powder with which it may be in contact.

With electric blasting caps the exploder may be placed either at the end or side of a cartridge. Care should be taken not to bend the wires sharply in any direction, owing to the danger of breaking the insulation, causing grounds, short circuits, and consequently misfires. The wires should be bound firmly to the cartridge by means of string, so that the electric blasting cap will not be displaced in loading or tamping. A great deal of trouble with misfires has been caused by taking hitches about the priming cartridges with the lead wires of an electric blasting cap or by bending them at sharp angles.

Position of Primer in Borehole.

When blasts are fired by means of blasting caps and fuse, there is quite a little controversy in some sections of the country in regard to the proper position in the charge for the primer. As far as the actual execution is concerned the position of the primer cuts no figure whatsoever except that it is best to have the detonator pointing toward the center of the charge. On the other hand the position of the primer is governed almost entirely by the kind of explosive and the quality of fuse used.

With the straight dynamites, owing to the great danger of igniting them, the primer should always be placed at the top of the charge. With the uninflammable ammonia dynamite it can be placed anywhere in the charge. With the gelatins the primer can be placed anywhere with small danger of ignition provided a very high quality of fuse is used.

The breaking qualities of the ground occasionally determine the placing of the primers. In the country east of the Rocky Mountains the common rule is to place the primer at the top of the charge, then leave enough room on top of the tamping to allow for the coiling of the excess of fuse.

In the Western States the general rule is to place the primer either at the bottom of the hole or just above the bottom cartridge. The miners in that section of the country state that if they should place the primer at the top of the charge the first holes would almost invariably blow the collars from some of the others cutting off the fuse, causing misfires and spoiling the round. By placing the primers at the bottom of the holes the fuses are all burning at a safe distance inside the collars when the first charges explode. The explosive most commonly used in that section of the country is gelatin and to lower the possibility of burnt charges to the greatest extent, nothing but the highest quality of gutta percha fuse is used. In regard to the priming of large charges, it might be well to mention that Monro says that it is essential to observe that explosive material does not detonate because it transmits the explosive wave, but on the contrary, because it arrests it. This means that a certain amount of the force of the original impulse is consumed, as the explosion progresses, by breaking the bonds that hold the explosive compound together. The amount of energy lost depends on the stability of the explosive compounds, the ease with which the explosive wave is transmitted and the presence of any interstices between the particles of the explosive substance. The rate of propagation of an explosive wave in a given substance increases with the density of the loading.

In the blasting of deep holes where the charges are of considerable length, the weakening of the explosive wave can sometimes be noticed by the failure of the charges to break bottom cleanly. In order to keep the explosive wave at a maximum, the common practice, in deep holes, is to use several strong blasting caps imbedded not over five feet apart in the charge, as well as some extra electric blasting caps. These shots are best fired from a power current with the electric blasting caps connected in parallel. The caps will then all fire simultaneously and thus serve the purpose of keeping the energy of the explosive wave up to its maximum throughout the charge. The having of more than one detonator in a hole is an extra precaution and can be considered in the nature of insurance against loss due to possible misfires or incomplete detonation of the long charge.

Incomplete Detonation.

According to the method employed for ignition, dynamite may be either slowly decomposed without ignition; or it may be burned briskly; or give rise to a moderate explosion capable of dislocating rocks or even locally crushing them; or it may be detonated to produce the most violent effects.

Dynamite made of pure ingredients undergoes practically no change when submitted to a temperature of 212° Fahrenheit for one hour. Heated rapidly it takes fire a little below 392° Fahrenheit the same as nitroglycerin. If ignited it burns slowly without exploding, but if enclosed in a space with resisting walls, it explodes under the influence of heating. The same thing takes place sometimes in the inflammation of a large amount of unconfined dynamite, owing to the progressive heating of the interior parts, which brings the entire mass to the temperature of spontaneous ignition.

As already mentioned, Berthelot states that combustion and detonation represent the two extreme limits of an almost infinite number of explosive reactions. Between these two limits an entire series of intermediate reactions are observed. The unlimited number of these reactions is demonstrated by the effects of the various methods of inflaming dynamite. This can also be shown by the influence of sufficiently strong tamping which can convert a simple inflammation into a true detonation.

This variety of phenomena is due to two orders of causes, one being mechanical and the other more particularly chemical.

From a mechanical point of view it is conceivable that between the two limits of progressive combustion and detonation, intermediate modes of decomposition may be produced, according to circumstances. That is, when combustion takes place, the local conditions surrounding the explosive substance, such as confinement, etc., have a direct influence with which this combustion may be changed into a true detonation.

The chemical phenomena also may vary according to the conditions under which the reaction may be brought about. In the detonation of explosive compounds it is essential that complete combustion should take place. This does not necessarily occur in slow inflammation, effected at low temperatures, in which incomplete reactions may at first take place. Thus, when dynamites and other explosive compounds, which have a complete "balance of formula," are burned or incompletely detonated, the oxides of nitrogen and carbon monoxide are given off in amounts which vary according to the heat developed by the reaction or reactions involved. In the proper detonation of these same explosives the gases would contain only nitrogen and carbon dioxide.

The following table, showing the percentage of different permanent gases at ordinary temperatures, after the solid and liquid products of the explosive reaction have been moved, is taken from the abstract of the report of the Western Australian Commission, given in the English "Blue Book" of the year 1905. The character of permanent gases after blasting is what interests the user of explosives as they show what can be expected in the faces and muck after blasting when the men return to work.

Explosives—	CO_2	CO	N_2	O_2	H_2S	NO	H_2	CH_4
Nitroglycerin exploded ...	63.2		31.5	5.3				
Nitroglycerin burnt	12.7	35.9	1.3			48.2	1.6	0.3
Blasting gelatin	68.5		31.5					
Gelignite	60.0		33.4	6.6				
Nitro cotton exploded	*21.7	49.3	16.3				12.7	
Nitro cotton burnt	41.9	5.8				24.7	7.9	1.3
Blasting powder	32.15	33.75	19.03		7.1		5.24	2.73

*Berthelot.

Gelignite (Guttmann)

Nitroglycerin	62.50
Nitro cotton	2.50
Sodium nitrate	26.25
Wood pulp	8.40
Soda	0.35
	100.00

Gelignite (Weiskopf)

Nitroglycerin	63.4
Nitro cotton	12.8
Sodium nitrate	14.2
Cellulose	9.6
	100.0

In regard to the gases from Gelignite, the 6.6 per cent. of oxygen represents the oxygen excess in all the high explosives formulae to take up the paper and paraffine of the cartridge. Thus if the paper is taken from the cartridges before loading, part of their strength is lost. Also this oxygen will, under those circumstances, be liberated as nitric oxide (NO) which is poisonous and adds to the bad gases liberated on detonation. It is well to note that the generation of poisonous gases by burning high explosives applies to all varieties and is not, as supposed by many, confined to those only which contain nitroglycerin.

It often happens that when a slow decomposition takes place certain gaseous products are evolved more copiously than others, with the result that the final detonation of the remainder of the explosive adds to, rather than diminishes the noxious vapors given off. Under these circumstances, the number of possible decompositions of any explosive substance is manifold, the reactions depending on the temperature, pressure and the quickness of heating.

Among the numerous modes of decomposition of a given explosive substance, those which develop the greatest heat are those which give the most violent explosive effects, all things else being equal. On the other hand, these reactions are not the ones that become manifest when the lowest temperature of decomposition is reached. If, therefore, an explosive body receives in a given time a quantity of heat which is insufficient to carry its temperature up to a degree which corresponds to its most violent reaction, it will undergo a decomposition which will disengage less heat, or even absorb it, and by this decomposition become completely destroyed without developing its most energetic effects. In short, the multiplicity of possible reaction involves a complete series of intermediate phenomena.

According to the mode of heating, it may happen that several methods of decomposition will succeed one another progressively. This succession of decompositions gives rise to even more complicated effects, as instead of causing a complete elimination of the decomposed part, it may result in the division of the primitive substance into two parts; one of which is gaseous becoming eliminated, and the other solid or liquid, which remains exposed to the conservative action of the heat. The composition of this residue being different from the original explosive substance, the effects of its consecutive destruction may become completely changed from those of the original explosive. This is shown by the solid residue left by burned dynamite.

Such are the causes, some mechanical and some chemical, owing to which nitroglycerin or other high explosives give such diverse effects, according to whether they are inflamed, heated, or exploded by means of a detonator charged with fulminate of mercury.

In regard to the relative efficiency of explosives when burned and when detonated, Roux and Serrau give the following results calling the effects obtained by means of gunpowder unity:

	Detonation.	Inflammation.
Nitroglycerin	10.0	4.8
Compressed guncotton	6.5	3.0
Picric acid	5.5	2.0
Potassium picrate	5.3	1.8

In practice, the fact that an explosive has not been properly detonated is made manifest principally by the production of considerable quantities of disagreeable and poisonous fumes, the presence of unexploded powder, the small amount of work done by the powder, and often with high explosives a section of the borehole in which the powder has burned is unaffected. As already mentioned, the bad fumes in incompletely detonated powder are due to the fact that its decomposition was effected at a temperature below that which corresponds to the most violent chemical action. The result is that instead of producing nitrogen and carbon dioxide in the gases from the explosion, both the poisonous nitric oxide and carbon monoxide are formed.

The weaker effect from imperfectly detonated powder is due largely to two causes; the lesser heat of formation of carbonic oxide gas and the heat absorbed in the formation of nitric oxide. According to Boyles Law, the pressure of a gas increases proportionately to the temperature. Thus when carbon monoxide and nitric oxide are present in the gases from an explosion, we should have and do get poorer results than when carbon dioxide and nitrogen are liberated.

Further investigation during late years has shown the presence of volatilized nitroglycerin in the fumes from burning or incompletely detonated dynamite. Nitroglycerin is very volatile and a small quantity may easily be evaporated by the heat from burning powder. This is made manifest by the action of these fumes on human beings. It is a very common fact that men, breathing the fumes from nitroglycerin explosives, in particular when improperly detonated, get violent headaches, similar to those due to slight nitroglycerin poisoning. This great similarity and the fact that the same treatment effects a cure in both cases, is accepted by quite a number of authorities as satisfactory evidence that nitroglycerin vapor is present in the fumes from poorly detonated or burning dynamite.

The causes of incomplete detonation are very numerous. It may be due to weak detonators, damaged or wet detonators, improperly made primers, spaces which probably contain dirt between the cartridges in the boreholes, contact between the fuse and the powder, causing the powder to be inflamed from the fuse, a displaced detonator, insensitive powder, frozen powder, etc.

When an explosive is somewhat insensitive the use of a stronger deonator will often overcome this difficulty. Some dry ingredient powders, and some containing ammonium nitrate, which are made into cartridges, have a tendency to become hard and insensitive. With these a kneading of the cartridges with the fingers will loosen up the powder, whereupon it regains its original sensitiveness. The effect of heat on the sensitiveness of an explosive has already been discussed.

The admission of a small amount of moisture to the charge of any detonator very appreciably reduces its strength. Thus it is fairly common in cases where ordinary or electric blasting caps are not stored or transported properly, for the detonators to lose so much of their strength that they are not capable of exerting the amount of energy necessary to develop the most violent reactions in the explosive with which it is used. Therefore, the greatest care should always be taken to keep all detonators in a dry place, away from all moisture.

Some explosives appear to get insensitive under the influence of great depths of water. As a matter of fact that is due to the great pressure which lessens the effective shock that the detonator would exert at ordinary atmospheric pressure. Therefore, for blasting in great depths of water such as are to be found in some deep drill holes, nothing less than a No. 8 (30.8 grain) detonator should be used.

EXPLOSIONS BY INFLUENCE.*

So far the discussion has covered the development of explosive reactions either from the point of view of their duration in a homogeneous system, all parts of which are main-another mode of propagation in explosives, this propagation in an equally homogeneous system which is fired directly by means of a body in ignition or by a violent shock. But the study of explosive substances has revealed the existence of another mode of propagation in explosives; this propagation taking place at a distance and through the medium of the air or of solid bodies which of themselves do not participate in the chemical change.

We now refer to explosions by influence, which hitherto have been suspected from certain known facts in connection with the simultaneous explosion of charges of high explosives separated at certain distances.

A dynamite cartridge exploded by means of a priming of fulminate will cause the explosion of cartridges in its vicinity, not only by contact and by direct shock, but even at a distance. An indefinite number of cartridges in a straight line or regular curve can also be exploded in this way.

The distances at which explosions by influence may be accomplished depend upon the medium in which the explosion is confined, the means by which the explosive is supported, the temperature of the air, and direction of the wind.

When suspended free in the air, the cartridges have an easy recoil thus diminishing the violence of the shock and making each successive cartridge less susceptible to being exploded by influence than when resting on a firm foundation. With a rigid support, the detonation will be propagated further than if laid on loose or free soil.

The writer has also found that the distance at which explosions by influence can be accomplished out of doors is very greatly affected by the intensity and direction of the wind; an explosion being obtained at a much greater distance when propagated in the same direction as the wind than when propagated at right angles to it, or in the opposite direction.

In the matter of varying degrees of confinement, experi-

*The Technical discussion of this section is from Berthelot.

ments made in Austria have shown that explosions by influence have been accomplished through open air at 4.0 centimeter (approximately 1⅝ inches) intervals, and through deal plank 1.8 centimeters (approximately ¾ inches) thick. In a lead tube 0.15 meter (approximately 6 inches) diameter and 1 meter (39.37 inches) long, a cartridge placed on one extremity will cause the explosion of another cartridge placed at the opposite end. The transmission of the explosion is more easily effected in tubes of cast iron. Joints lessen the susceptibility of transmission.

The explosion thus propagated may grow weaker from one cartridge to another, and even change its character. It has been found that when a line of cartridges, with air spaces between them, is exploded by means of a blasting cap placed in a cartridge at one end of the line, the craters formed by the explosion grow smaller according to the distance from the primer.

According to these facts propagation by influence depends both on the pressure acquired by the gases and on the nature of the support. It is not necessary that this support should be firm. It has been ascertained that these effects are not generally due to simple projections of fragments of the casing or neighboring substances, although such projections often play a certain part in the phenomena.

In this respect the real character of the effects produced is shown more particularly from tests made under water. In fact, when experimenting in water below a depth of 1.30 meters (4.28 feet) a charge of dynamite weighing 5 kilograms (11.0 pounds) will cause the explosion of a charge of 4 kilograms (8.8 pounds) at a distance of 3 meters (9.84 feet). The water, therefore, transmits the explosive shock, to a certain distance, in the same way as a solid body. Sudden pressures transmitted by water have been measured for distances up to 5.50 meters (18.04 feet) by means of a lead crusher. These pressures decrease with the distance as might be expected. Experience has proved that the relative position of the charge and the crusher is immaterial, this being in accordance with the principle of equal transmission of hydraulic pressures in all directions.

It follows from these facts and particularly from experience made under water, that explosions by influence are not due to inflammation properly so-called, but to the transmission of a shock resulting from enormous and sudden pressure produced by the explosion of nitroglycerin, guncotton or other high explosives, the energy for which shock is transformed with heat in the explosive substance which is placed at some distance from the first explosive.

In an extremely rapid explosive reaction, the pressure may approach the limit which corresponds to that of the matter exploding in its own volume; and the disturbance, due to the sudden development of pressures, nearly equal to the theoretical, may propagate itself either by the mediation of the ground and of the supports, or through the air itself, as has been shown by experiments made with dynamite, compressed guncotton, etc.

The intensity of the shock propagated either by a column of air or by a solid or liquid mass varies according to the nature of the explosive body and its mode of inflammation; it is more violent the shorter the duration of the chemical action and the more gas there is developed, or in other words, the quickness and strength.

This transmission of shock is more easily effected by solids than by liquids, and more easily by liquids than by gases; in the case of gases it takes place more easily if they are compressed. It is propagated more easily through solids when these are hard; iron transmits shocks better than earth and hard earth better than soft soil. Any kind of a junction has a tendency to weaken, especially if a softer substance intervenes.

Explosions by influence propagate themselves all the more easily in a series of cartridges, if the casing of the first detonating cartridge is very strong. This allows the gases to attain a very high pressure before burstng the casing.

The existence of an air space between the fulminate and the dynamite will, on the other hand, diminish the violence of the shock transmitted, and consequently that of the explosion. As a general rule the shattering effect of dynamite is lessened when there is no contact between the cartridges.

In order to form a complete idea of the transmission by supports of sudden pressures which give rise to shocks, it is well to bear in mind the general principle whereby pressures in a homogeneous medium transmit themselves equally in all directions and are the same over a small surface, whatever may be the direction that the surface in question faces.

The results of experiments in the transmission of explosives under water have shown that this principle is applicable to sudden pressures produced by explosive phenomena, but this ceases to be true when passing from one medium to another.

If the chemically inactive substance which transmits the explosive movement be fixed in a given position on the ground, or on a rail on which the first cartridge has been placed, or again held by the pressure of a mass of deep water, in which the first detonation has been produced, the propagation of the movement in the matter could scarcely have taken place except under the form of a wave of a purely physical order, a wave, the character of which is essentially different to the first wave which was present at the explosion, the latter being both of a chemical and physical order, and having been developed in the explosive body itself. While the first or chemical wave propagates itself with a constant intensity, the second, or physical wave, transmits the vibration starting from the explosive centre and all around it, with an intensity which diminishes in an inverse ratio to the square of the distance. In the immediate neighborhood of the centre, the displacement of the molecules may break the cohesion of the mass, and disperse it, or crush it by enlarging the chamber of explosion, if the experiment be carried out in a cavity. But at a very short distance from the point of the explosion the greatness of this displacement depends on the elasticity of the surrounding medium. These movements, confused at first, regulate themselves, so as to give rise to the wave, properly so-called, characterized by sudden compressions and deformations of the substance. The amplitude of these undulatory oscillations depends on the greatness of the initial impulse.

These physical waves travel with a very great rapidity, at the same time constantly decreasing in intensity, and they maintain their regularity up to points at which the medium in which they exist is interrupted. Then these sudden compressions and deformations change their nature and transform themselves into an impelling movement; that is to say, they reproduce the shock. If they act on a fresh cartridge they will cause it to explode. This shock will be further attenuated by distance owing to the decrease in intensity of the waves.

Consequently the effects of the shock may be modified to such an extent that the character of the second explosion may be changed even to simple inflammation. These effects will be thus diminished until a certain distance is reached from the point of origin beyond which distance no results will be obtained.

When the explosion has taken place in a second cartridge, the same series of effects is reproduced from the second to the third cartridge, but these will depend directly on the character of the explosion in the second cartridge and so on.

Such is the theory which, according to Berthelot, accounts for explosions by influence, and for the phenomena which accompany them. It rests on the production of two orders of waves, the one being the explosive wave, properly so-called, developed in the substance which explodes, and consisting of a transformation increasingly reproduced from chemical actions into calorific and mechanical actions, which transmit the shock to the supports and to contiguous bodies; and the other purely physical and mechanical which also transmits sudden pressures around the centre of vibration to neighboring bodies, and by a peculiar circumstance to a fresh mass of explosive matter.

The explosive wave once produced, propagates itself without diminishing in force, because the chemical reactions which develop it regenerate its energy proportionately along the whole course, whereas the mechanical wave is constantly losing its intensity in proportion to its energy, which is determined only by the original impulse, and is distributed into a more considerable mass of matter.

The practical applications of explosions by influence are rather limited. The best known case is the propagation method for blasting ditches. This consists of blasting a row of holes by means of a heavy primer at the center. This can only be accomplished when there is sufficient ground water to cover the charge in each hole. The only explosives that can be used successfully are the regular 50% and 60% nitroglycerin dynamites, which are the most sensitive explosives on the market. The maximum spacing on record under the most favorable conditions is 30 inches. The material in this case was liquid muck. The recommended distance is not to exceed 18 inches. The water transmits the shock from one charge to the next and if no ground water is present this method of blasting will be a failure.

Once in a while in underground work a miner will be found who makes use of explosions by influence in distributing a charge in a bore hole. In this work the depth of a hole rarely exceeds 7 feet. Part of the charge is placed in the bottom of the hole, then a stick of wood about a foot long is inserted and the remainder of the charge placed on top of this. Sometimes the charge may be broken in two or three places but never more than this. In order to be sure of the results, the stick must be absolutely free from any binding in the bore holes and each end must be in contact with the dynamite. Hardwood will transmit the explosion better than soft. An iron or steel rod will be still better. Care should be taken that the stick does not slip past the dynamite at either end.

We have already seen that the shock causing an explosion by influence is greatly modified when passing from one medium to another. In a bore hole if the cartridges of dynamite are not in close contact, provided there is no foreign matter present, the explosion will be complete throughout the charge. However, it takes a very small amount of dirt between the cartridges to affect the physical wave or shock to such an extent as to change true detonation to incomplete detonation or even simple inflammation or combustion. In turn simple inflammation produced in this manner in the confinement of a bore hole may produce enough heat to cause a recurrence of true detonation, giving two distinct reports to one blast. An occurrence of this kind is very rare and is usually accompanied by the production of very bad fumes.

Another method of applying explosions by influence is in blasting holes for telegraph poles, etc., in hard pan. In this case small charges of dynamite are tied at intervals to a stick, a blasting cap being placed in the top cartridge. The length of this stick is the same as the depth of the hole. The primer should be at least one foot below the collar of the hole. The object of distributing the charge in this manner is to excavate a proper sized hole without disturbing the ground which will have to support the pole after it is put in place. Another method of accomplishing this is to fasten the dynamite at proper intervals in a heavy paper or pasteboard tube.

Velocity of Detonation.

The velocity of detonation or quickness of an explosive is often as important a point to consider as its strength in the selection of dynamite for any work in particular. The greater the rate at which an explosive is decomposed on detonation, the greater will be the crushing and shattering effects. According to Berthelot, there is a direct relation between the sensitiveness of a high explosive and its quickness or rate of detonation. A powder that is sensitive is easy to detonate and the explosive wave, being propagated with ease, gives the sensitive powder a higher velocity than is the case with an explosive not so sensitive to detonation.

In explosives containing nitroglycerin, the quickness depends on the actual amount of nitroglycerin contained, the other ingredients with which it is mixed, and the impulse with which it is detonated. The straight dynamites increase in quickness directly with the percentage of nitroglycerin.

The ammonia dynamites, as made for commercial use, increase in quickness in proportion to their strength, but their quickness does not necessarily increase with their strength because an increase of ammonium nitrate, the nitroglycerin remaining the same, will increase the strength of the explosive but at the same time reduce its quickness. The principle of the effect of the relative amounts of nitroglycerin and nitrate of ammonia is of great value in the manufacture of special explosives for certain classes of work sufficiently large to warrant a departure from the regular formulae.

The gelatins from 35% to 80% strength being quite insensitive are all comparatively slow in their action under an ordinary fulminate detonator, but under the influence of a primer of straight dynamite of 40 or greater percentage strength of blasting gelatin their action becomes quicker than the corresponding grades of straight dynamite.

Blasting gelatin is the quickest as well as the strongest explosive on the market at the present day. In tight blasting, good results are often obtained by placing one or two cartridges of a very strong explosive at the bottom of each hole. When this is done care should always be taken to use a dynamite that is as quick or quicker in its action at the bottom of the hole as the one loaded above it. If a slow explosive is used at the bottom, the quicker one above will open up the rock so that part or all of the effect of the slow explosive below is lost.

Efficiency of Explosives.

(Condensed from C. E. Bichel—"Testing Explosives").

The exertion of chemical activity in a bore hole on explosion of a blasting charge is obviously beyond the scope of immediate observation, as the conversion of explosives into gases is too rapid for scrutiny. In order to get an insight into the probable happenings attending an explosion, one is therefore compelled to picture the sequence of events with a view towards applying such means of observation as may be applicable to certain points or stages.

In blasting operations, a properly prepared borehole is charged with cartridges of a suitable diameter, which are inserted one by one and gently pressed home so as to insure perfect contact with the walls of the borehole. The last cartridge is primed with a squib or fuse if it be black powder; or, if it be a high explosive, a blasting cap and fuse, or an electric blasting cap, will be used; the primer is pushed in until it touches the top cartridge; and the remainder of the hole is firmly stemmed with good tamping material. To obtain the maximum effect no empty space should be left in the borehole and the stemming should be as firm as possible to insure the maximum resistance against the pressure of the gases of explosion.

On firing the shot, the flame of the powder fuse—or in the case of electric blasting caps, the current—ignites and explodes the fulminating composition in the detonator. The intensely hot products of decomposition of the fulminate strike into the first cartridge, causing it to detonate and, in its turn transmit the explosion to the next cartridge and so on from cartridge to cartridge. The whole of the charge is thus converted into new combinations either gaseous, solid or liquid as the case may be, which immediately strive to occupy an increased space due to their gaseous transformation and to the further potential expansion derived from the heat produced by the explosion. If the shot does its work, the resulting gases shatter the material surrounding the borehole, and escape through the broken mass. The mechanical work thus expend-ed and the subsequent contact with the surrounding air cause the gases to cool and ultimately diffuse.

The question is "By what means can these phenomena be gauged?" Apparatus is designed for the purpose of measuring with sufficient exactness the lapse of time between the detonation of the first and last cartridge. It is this measure of the "rate of detonation" which enables us to gauge the dynamic energy of the products of explosion. It is the velocity at which the superheated gases, during and after formation, acquire a displacement of frequently over a thousand times the volume originally occupied by the explosive, and at which they are projected against the resisting walls of the shot-hole with disintegrating, scorching, melting and even vaporizing effects. Both the gases and vapors must necessarily fully participate in this kinetic energy generated by the detonation, while the more inert and solid products of decomposition, though doubtless acting as projectile bodies, are not impelled with the same force owing to their lack of expansion. Nevertheless, in computing the total converted energy of a given explosive, the whole of its constituents must be considered, as it is only reasonable to assume that such products as may be solid at ordinary temperatures will, in the majority of cases, be molten or even vaporized at the actual moment of explosion.

The energy imparted to the products of decomposition by the detonation may be expressed thus:

$$\frac{Mv^2}{2}$$

M representing the mass of the decomposition products and v the rate of detonation.

The calculated value of this kinetic energy represents the "percussive force" of the explosion.*

But a further factor must be taken into account, viz., the pressure developed by the expansion of the superheated gases. Explosion temperatures are, as a rule, calculated on the basis of the total heat developed by given quantities of the different explosives, their specific heats and the products of combustion

*This depends upon the assumption that the observed rate of detonation is identical with the velocity of molecular projection. This assumption is now agreed upon by almost all authorities on explosives.

determined by analysis. Adopting the temperatures so calculated and assuming that Gay-Lussac's law for the expansion of gases

$$V = V_0 \left(1 + \frac{t}{273}\right)$$

holds good at such temperatures, the pressure exerted by the heated gases and vapors can be deduced. While the percussive force is a *dynamic* action, expressible in kilogram-meter-seconds, the pressure, due solely to the thermo-expansion of the highly compressed gases and vapors, must be considered as *static* energy, and is therefore expressed in kilograms per square centimeter or pounds per square inch.

If, for instance, gunpowder be compared with brisant (detonating) explosives, experience shows the former to be greatly lacking in percussive force. So much so, in fact, that if a powder charge be fired from a bore in hard tough rock it is blown out as from a gun barrel without shattering or even perceptibly affecting the borehole. It is this very deficiency in percussive force which renders gunpowder a suitable propellant for use in guns; high explosives with violent percussive force would burst them.

That the brisancy (smashing effect) of high explosives varies in degree, according to the relative rate of detonation is confirmed by experience of the miner, who finds that the harder and tougher the rock the quicker and stronger must be the explosives. A suitable agent to employ under such conditions is blasting gelatin; this can be used in cartridges of small diameter (diminishing boring costs) due to the fact that it requires a smaller space in which to develop a high rate of detonation than do explosives detonating at a slower rate. Again, explosives of the latter class (more particularly those of the ammonium nitrate type, the pressure of which never reaches that developed by high percentage nitroglycerin compounds) are found advantageous mainly where the rock surrounding the borehole is easily shattered by the heat of the explosion, quickly crushed by the percussive force and readily disintegrated by the gas pressure. Here a more violent explosive would obviously be of little use, for while the percussive force would instantly create an increased space, neutralizing the advantage of small charging density, the final gas pressure would also be lowered by the quicker condensation of the products of decomposition on contact with the larger surface area produced. Miners speak of such a shot as "having killed itself" (American term "pot holing") perhaps an appropriate description of what actually takes place.

If the object of blasting were merely to crush or shatter, then, no doubt, violent explosives would serve best, but as practical mining rather involves the shifting and getting of masses, it stands to reason that this will be often accomplished more easily and economically by using a less violent agent, i. e. one with a lifting or heaving action. In soft ground again, it is pressure which must mainly accomplish the work and as highly percussive explosives would interfere with this action, they are unsuitable. In short, quickness is as important as strength in selecting a proper explosive for blasting.

The Use of Blasting Powder to Detonate Dynamite.

Berthelot states that black powder will explode nitroglycerin but will not detonate dynamite. Under ideal conditions straight nitroglycerin dynamite can be detonated by black powder provided the confinement be sufficient. However, this is strictly an explosion caused by heat and a certain amount of dynamite must necessarily be burned before enough heat is generated to explode the remainder.

There are a number of places in the East where blasting powder and dynamite are used in the same borehole. The best method is to place the dynamite in the bottom of the blasting powder cartridge with an open blasting cap placed in the end of the last stick of dynamite. The cartridge is then filled with black powder. Some load the dynamite in the borehole with an open cap in the last stick and then place the blasting powder cartridge on top, but care must be taken that no dynamite gets into the open cap, otherwise the nitroglycerin may "kill" the fulminate. The hole is tamped well and the charge fired by igniting the blasting powder by any of the ordinary methods.

The idea of this practice is that practically all the black powder is burned before the fire reaches the blasting cap and explodes the dynamite. Thus, after the black powder has burned and exerted its full explosive force, the dynamite explodes giving an extra "kick" and cuts out everything cleanly to the bottom of the hole. This is particularly efficacious where there is a heavy burden on the point of the hole.

This method is used principally in the brick clay mines of Western Pennsylvania and Eastern Ohio. It is not considered as safe as it might be and is not as a rule recommended by the different manufacturers of explosives.

DYNAMITE AS A PRIMER FOR BLACK POWDER.

Dynamite is used almost exclusively, at the present time, for exploding large charges of black powder. The advantages of this practice are, first, it is easier to place the igniter in the center of a charge and there is less danger of its being displaced when a cartridge of dynamite is used; second, the heat, flame and pressure from the explosion of a cartridge of dynamite causes a much more rapid rate of burning and therefore a quicker and more violent action from the black powder. This practice also assures good results when a line of holes loaded with black powder is fired simultaneously by electricity. In very large charges two or more primers can be electrically connected and used to very good advantage.

ELECTRIC BLASTING.

Blasting by electricity, where it can be used, has many advantages over cap and fuse as regards safety, saving in time, and work accomplished. All the holes in a round may either be exploded simultaneously by Instantaneous Electric Blasting Caps, or charges may all be set and primed at one time but made to fire in rotation either separately or in groups by the use of Delay Electric Blasting Caps, also called Delay-Action Exploders.

By the instantaneous electric method, all holes are exploded simultaneously and there is no possibility of a second explosion. Any misfire whatever the cause may be, will not hang fire as sometimes happens when the blasting is done with fuse and caps. A line of holes or set of holes fired simultaneously does much greater execution than would be the case if these holes were fired singly.

Where it is necessary to have certain holes explode in proper rotation, as in shaft sinking, also drifting in heavy ground and similar work, this can be accomplished by delay electric blasting caps, or delay-action exploders, such as are on the market today. These devices consist of fuses of varying length, having a blasting cap at one end and an electric igniter at the other end, made up in one unit properly protected against water, and convenient to handle. The shots at each period of delay do not explode exactly together, but they are timed so that one period will not overlap into the next period, and it is therefore feasible to maintain the proper rotation. The principal features of these delay-action exploders are safety, time saving and effectiveness in places difficult of access and egress, such as in shafts, winzes, raises, etc. The abence of smoke is another advantage in the use of delay-action exploders in poorly ventilated places.

There are three general methods of making connections for electric blasting. They are "series," "parallel," and "parallel-series."

Series connection is made by taking one of the wires from the first hole and connecting it with one of the wires from the second hole, then taking the other wire from the second hole to one of the wires from the third hole, and so on until the last hole is reached; the remaining wire from the last hole is connected to one of the leading wires from the blasting machine and the free wire from the first hole is connected to the other leading wire. Series connection is necessary when firing with the usual push down type of blasting machine, and often with electric lighting or power current.

The current required for firing in series connections is 1 ampere and voltage enough to overcome the resistance of the sum of all the electric caps in the series. Resistance varies with the length of wires, about one volt being required for each electric cap up to 12 ft. wires; too high voltage often causes misfires from short circuits across the cap wires without heating the bridge, especially when more than one blasting cap is used in the same bore hole.

Parallel connection is made by connecting one wire from each cap to one leading wire and connecting the second wire from each cap to the other leading wire. This method can be used only where enough current is available to supply one ampere for each cap connected in the parallel circuit. Thus, if there are 20 caps all in parallel the firing current must have 20 amperes and the voltage may be anywhere from one volt to one thousand volts. Parallel connection must be used wherever the firing current is to be taken from a heavy power or lighting circuit. If it be a lighting circuit, it will be well to connect into the firing line a fusible fuse block having amperage of capacity a little less than that in the fuse to the lights, so that if the blast short-circuits any of the wires, the fuse in the firing line will blow and thus prevent blowing out the lights.

Parallel-Series connections are made by dividing the total quantity of caps to be fired into groups, each group having the same quantity of caps. The caps in each group are connected in series; and the groups are connected in parallel to the firing line by attaching one of the two free wires from each group to each of the leading wires. The success of this method depends upon an even distribution of the current, which can only be accomplished when the resistance of each series is the same as of the others; about the same length of wire should be used on every blasting cap and the number of caps in each series should be the same. This system is used when necessary to fire a larger number of charges than there are amperes in the current; the amperage available being first ascertained, will determine the maximum number of groups because each group will require one ampere. Then divide the total quantity of caps by the number of groups to ascertain the voltage required to overcome the resistance of all the caps in each group, which will roughly amount to as many volts as there are caps in one group. The number of groups does not affect the voltage, and any voltage in excess of the quantity required does not count. For example: 50 caps connected in Parallel-Series of 5 groups in parallel with 10 caps each in series, would require 5 amperes and at least 10 volts.

Direct current is best to use whenever possible, but alternating current is equally efficient when of a frequency of 60 cycles or more, and can be used down to 30 cycles. Alternating current of lower frequency may cause trouble from misfires of the less sensitive electric blasting caps in the circuit when connected in series.

No electric blasting cap fires in less than 0.014 seconds. With a 60-cycle alternating current 2 alternations occur in 0.0167 seconds. Below 25 cycles (complete alternation in 0.02 seconds) trouble may be encountered, due to the building up effect.

In making any kind of electrical connection, the ends of the wires should be scraped bright and clean, and be twisted tightly together. Bare wires or connections must be kept from touching the ground at any place whether wet or dry; and if water is present all connections and other bare parts of the wires should be covered with insulating tape. The leading wires should be bent in to a hook at the end to prevent the smaller circuit wires from slipping off should the leading wires be subjected to strain.

A special type of galvanometer is available for testing the circuits to detect breaks, bad connections or short circuits after connections have been made and before firing. The galvanometer for this purpose is to be attached at the safety ends of the leading wires, that is, the ends which are to be attached to the firing switch or blasting machine.

Leading wires should never be connected to the blasting machine until everything is ready and everybody is away from the face of the blast; then, when the leading wires are inserted into the binding posts of the machine, they should be firmly secured by the thumb nuts. The blasting machine should be placed on a level spot (dry board or plank is best) to prevent it tipping over, and the handle should be operated with both hands full force.

If firing is to be done by means of a power or lighting circuit a special switch should be used, of such design that it can be seen at a glance whether the circuit is open or closed. Switches of complicated construction, especially those having springs, should be avoided. The switch should be so constructed that it can be locked in the open position. The cutouts on the switch should be of ample capacity.

Dry cells are sometimes used to fire blasts, but these are not recommended for firing more than one electric blasting cap at a time. All electric blasting caps are tested within very close limits for their electrical resistance before being placed on the market, but it is impossible to have the resistance absolutely identical for all. When firing with a blasting machine or dynamo the current is built up to its full strength before it is diverted through the blasting circuit, thus overcoming any slight variations; but when a dry cell is used the current necessarily starts to build up from nothing. The result is that often when more than one electric blasting cap is in the circuit, one cap a little more sensitive than the others will explode first and cause the remainder to misfire.

The electrical conductivity of ground waters sometimes makes trouble for electric blasting because very small percentages of certain salts, such as alkaline chlorides, soluble salts of copper, etc., materially increase the conductivity of water and often cause misfires by excessive leakage of current. This was very noticeable on the Panama Canal where special waterproof insulated wires had to be used. This is something that should be borne in mind at all times, as waterproof electric blasting caps have sometimes to be used on comparatively dry work in order to insure the explosion of all shots when exposed to the above influences.

Detonating Fuse.

Detonating fuse, often called by the French name, "Cordeau Detonant" consists of a lead tube filled with trinitrotoluene, (commonly called "T. N. T.") This has an extremely rapid rate of detonation and can be used to increase the shattering effects of practically all grades of high explosives. Detonating fuse is sold in lengths from 100 to 500 feet and where necessary its tensile strength is increased by a tight cord winding, called countering, on the outside.

It must be detonated by means of an electric or ordinary blasting cap placed so that the end is in close contact with the T. N. T., which is accomplished by a special brass sleeve.

Detonating fuse can be used for blasting deep well drilled holes in quarries where column loading is used and a better shattering action is desired. It can be used for blasting tight cuts in hard rock tunnels and as a substitute for electric blasting where it is necessary to blast an extremely large number of holes at one time, and there is insufficient current available.

In loading a bore hole a length of detonating fuse sufficient to reach from the bottom to a distance outside to allow for connections, is placed in the borehole. In deep well drilled holes the countered variety should be used. Then the hole is loaded and tamped in the regular manner. After the charging is finished, a line of the plain variety is laid along the collars of the bore holes. The ends of the lengths coming from the individual bore holes are then split for 3 or 4 inches. A special tool for this purpose is furnished by the manufacturers. After slitting separate the legs and then placing the main line snugly in the crotch, wind the legs about it, one to the right and the other to the left. As T. N. T. is extremely insensitive care must be taken that the joint is tight and that the main line is in close contact with the material exposed in the crotch and that the joint is absolutely dry. The branch lines should be at right angles to the main trunk for the first 3 or 4 inches. At the end of the main line the detonator is fixed to fire the blast.

The action of detonating fuse is that of a detonator extending through the entire length of the charge and the result is that no matter what explosive is used its action must be just as quick as that of the detonating fuse passing through it. The economy effected consists in breaking the material smaller thus reducing the cost of subsequent block holeing and mud capping. Detonating fuse is not adapted for use where charges are loaded in sprung holes but only where column charging is used.

Auxiliary Detonators—"Boosters."

In order to insure the maximum possible effect from high explosives an auxiliary detonator called a booster has recently been placed on the market. This consists of a brass tube about 5 inches long containing a length of detonating fuse. An opening at one end is left long enough to admit a blasting cap.

To use them with ordinary blasting caps, the cap is crimped on the fuse in the regular manner and then placed in the open end *in contact* with the detonating fuse inside and then the brass tube is crimped on the fuse above the cap. With electric blasting caps the cap is placed in the same manner and the tube crimped about the wires. The booster is then placed in the cartridge which becomes the primer.

Boosters are made in different sizes for use with the various ordinary and electric blasting caps.

So far they have not been used to any large extent. The writer's experience has been that practically absolutely complete detonation is shown by the absence of fumes after blasting and a better execution, particularly in hard tight work. Boosters should be very effective in tunnels, drifts, shafts, raises and other tight work, particularly where the ventilation is poor. They are particularly effective with gelatins.

Conclusions.

In conclusion, it has been shown that the use of strong detonators is of the utmost importance in order to secure good results. This fact is recognized in Europe to be of such importance, that with the exception of their dynamite No. 1 (75% nitroglycerin, 25% keiselguhr) the use of any detonators of a strength less than No. 6 is prohibited by law. Strong detonators overcome many of the possibilities of misfire, burning powder, and improperly detonated powder due to displaced cap, or spaces between cartridges in a borehole, or to insensitive powder, etc. Therefore, by insisting upon stong blasting caps, the proper detonation of the charge is assured.

All explosive substances become more sensitive as their temperatures approach that of decomposition. Walke sums this up by saying, "All nitroglycerin preparations, when gradually heated up to their exploding points, become extremely sensitive to the least shock or blow."

As a rule, explosives that are quickest in their action are the most sensitive. The admixture of foreign matter lowers the sensitiveness of an explosive compound in proportion to the amount of this foreign matter that is used.

Explosives containing liquid nitroglycerin are the most sensitive. Gelatins are less sensitive than those containing liquid nitroglycerin and dry ingredient powders are the most insensitive of all. But, although an admixture of dry ingredients affects sensitiveness, many of these ingredients increase the strength and improve the gases of the explosive.

It is possible to manufacture explosives the gases from which are not poisonous but it is impossible to have an explosive whose gases will sustain life. The detonation of almost all explosives on the market today, when complete, produces hardly any injurious vapors but when they burn or are incompletely detonated they produce over 80% of mixed nitric oxide and carbonic oxide, which are poisonous.

In regard to quickness: The consumer should always remember that the quickness is about as important as the strength when determining the proper explosive for blasting certain material. The sensitiveness of an explosive to detonation has a direct bearing on its velocity of detonation. When a primer of 40% or stronger dynamite is used with gelatin, their quickness exceeds the corresponding straight grades by about 10%. This can be used to good advantage in tight blasting in hard rock, the density of the gelatins also being considerable aid. The gelatins are also best adapted for blasting where the water conditions are unusually severe.

Black powder alone will not detonate dynamite completely. It is often used, although not recommended, in connection with a cap properly placed to insure the complete detonation of of the dynamite when the latter is desired in the bottom of the bore hole. On the other hand, dynamite can be used to advantage as a primer for large blasts of black powder.

The question of handling explosives can be summed up as follows: Keep powder and detonators apart until they are ready to be used; keep the powder, fuse and caps dry; always thaw nitroglycerin slowly at moderate temperatures preferably not to exceed 80° Fahrenheit, with the cartridges lying on their sides; in short, "Be careful—Use every-day common sense."

RECORD SINKING AT THE HOMANSVILLE SHAFT OF THE CHIEF CONSOLIDATED MINING COMPANY, TINTIC DISTRICT, UTAH.

BY WALTER FITCH, JR., EUREKA, UTAH.*

The Homansville shaft of the Chief Consolidated Mining Company in the Tintic District, Utah, was begun on July 8th, 1916, and in the first thirty-one days was sunk a total of 256.3 feet. In the succeeding thirty-one days of August, the shaft was sunk a total of 261 ft., a record which has never been exceeded in any shaft.

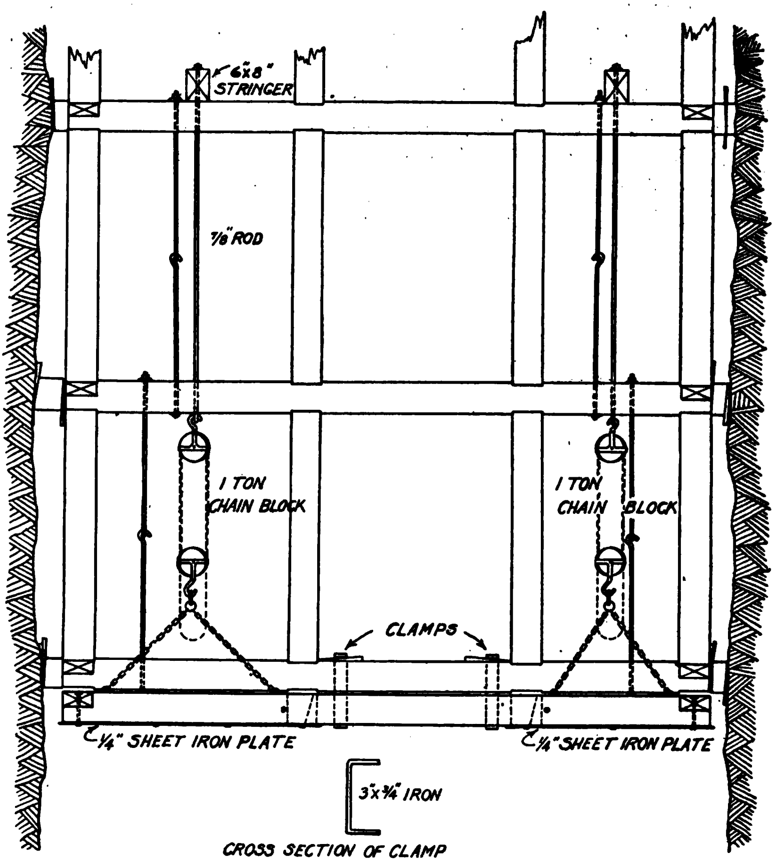
One of the interesting details of the method that was used in this work is what is termed "the suspended bulkhead and shooting set." These are shown in the illustration. This device permitted the timbering to be done without the stopping of the work in the bottom of the shaft. The bulkhead was suspended by two one-ton chain blocks from sets immediately above the bottom of the shaft and lowered just far enough to allow timbers to be installed. This suspended platform served as a stage for the timbermen to work upon and prevented anything from dropping on the men below, thus adding greatly to safety and to speed in sinking.

The outside dimensions of the shaft timbers are 14 ft. by 5 ft. 6 in. The shaft has two hoisting compartments, 4 ft. 2 in. square and a manway 4 ft. 2 in. by 3 ft. 2 in., all inside measurements. The timbers used are 8 by 8 in. throughout and are placed on 5-ft. centers in the porphyry formation and 6-ft. centers in the limestone. The shaft is lined throughout with 2 by 12-in. lagging.

The material in which the shaft was sunk was porphyry and limestone. The porphyry frequently gave trouble by sticking and plugging the hollow drills used; but on the other hand it always broke up very fine and was therefore easy to handle. The limestone was a blocky, close-grained rock of about aver-

*Contractor Mine Development and Tunneling.

age hardness. Either three or four small sinking drills were used with ⅞-in. hollow hexagon steel. A 5-ft. round of from 20 to 30 holes was drilled and shot every twelve hours. As the compressor was not large enough, the air pressure was always poor, ranging from 60 to 70 pounds. Hoisting was



SUSPENDED BULKHEAD AND SHOOTING SET
USED IN SINKING SHAFT

allowing the bucket to tip and discharge its contents on an done with two 15½-cu. ft. top-swing buckets, used alternately, which dumped automatically on top into a car. The dumping mechanism consisted of a fixed chain hooked into a ring in the bottom of the bucket; this held the bottom stationary, incline door and chute. The center compartment of the shaft was lined on the inside with lagging. The bucket was thus allowed to swing free in the compartment.

Three shifts of four machinemen each were used in the operations, drilling or mucking, as the case might be. All the timbering was done on day shift by two timbermen, with the occasional help of the foreman in charge. On top, the force consisted of a topman and engineer on each shift. Every man on the job received $5.00 per day, except topmen, who were paid $3.25 per day. The powder used was 1-in. 35- and 50-per cent. Extra Gelatine with number 8 caps.

The firm of Walter Fitch, Jr., incorporated, of Eureka, Utah, held the contract for the work.

SIGNALLING SYSTEM AT BENGAL MINE, PALATKA, MICHIGAN.

BY A. H. MACGREGOR, PALATKA, MICH.*

One of the most important details of any mine hoisting equipment is an efficient signalling system. This is true both from an operating and from a safety standpoint. It matters not how well designed or elaborate a hoisting equipment may be, the instant the signalling apparatus fails to do its work, confusion begins, and time is lost and tonnage is lost. From the standpoint of safety, no comment is needed, for everyone knows the danger to life and property from misunderstood or improperly transmitted signals.

Any signalling system, to be efficient, must meet the following requirements:

(1) It must transmit signals instantly to the place intended.

(2) The person giving the signal must know that it is correctly interpreted.

(3) The signalling apparatus must do its work with the minimum amount of attention and repair.

With these requirements in mind, the writer designed and installed a signal system for the Bengal mine, No. 1 shaft. At this property, the ore is hoisted from the second level approximately three hundred feet below surface to a crushing plant in the headframe eighty feet above the ground. Two three-ton self-dumping skips are used, these being operated in balance by a single-drum motor-driven hoist located 250 feet from the shaft. The ore is dumped from the skips upon grizzly bars inclined at 45 degrees. The lumps pass to the crusher and thence to a pocket and the screenings fall directly into the pocket. Just below this pocket, at a distance of 35 ft. above ground, is the landing where the grading and distributing is done. From this point the waste material is trammed to the

*Mechanical and Electrical Engineer, Verona Mining Company.

dump, and the ore runs through to railway cars on the track below, or goes to the stockpile, according to requirements.

The shaft has three 6 by 6-ft. compartments, two skipways and an emergency ladderway. Of the two underground stations, the one at the first level, at 200 ft. below surface, is used only for supplies, the second being the main loading station. The ore is dumped from saddle-back mine cars into short chutes, which discharge directly into the skips at a point 8 ft. below the main level floor. There are no storage bins at this point and each car must be spotted directly over the proper chute and tripped after the skip is in the right place to receive the material.

When the mine first started to produce, one shaft was used for the men as well as for materials, the cage shaft being not yet completed. For the men a small cage was used in place of one of the skips. Under these conditions, a more elaborate signalling system was needed than would have been required if only ore was to be handled. The problem was solved by treating this shaft as though it was a man-cage shaft, with additions to take care of the hoisting of ore.

Since there was a 110-volt 60-cycle lighting circuit on tap at all times, it was decided to adopt apparatus to use this current directly without further transformation, thus saving batteries and motor generators. The general lay-out of the whole system is shown clearly in the wiring diagram drawing. This drawing will show that seven distinct and separate signalling units are combined in one system. Switches at the two underground stations and the surface station operate separate bells on the engineer's platform for each of the two hoisting compartments. On the engineer's platform are two switches which operate bells simultaneously at all stations, each switch for its own compartment. On each side at the surface and underground stations, switches are provided which ring in unison all of the station bells, the same as when these are operated by the switches on the engineer's platform. After a signal is given to the engineer from any point, he answers the same exactly as he interprets it, using the switch which corresponds to the bell rung, before moving the cage. Having thus received the return signal, the man underground knows that his signal has been correctly delivered and understood by the engineer.

In the headframe there are two stations, one at the landing

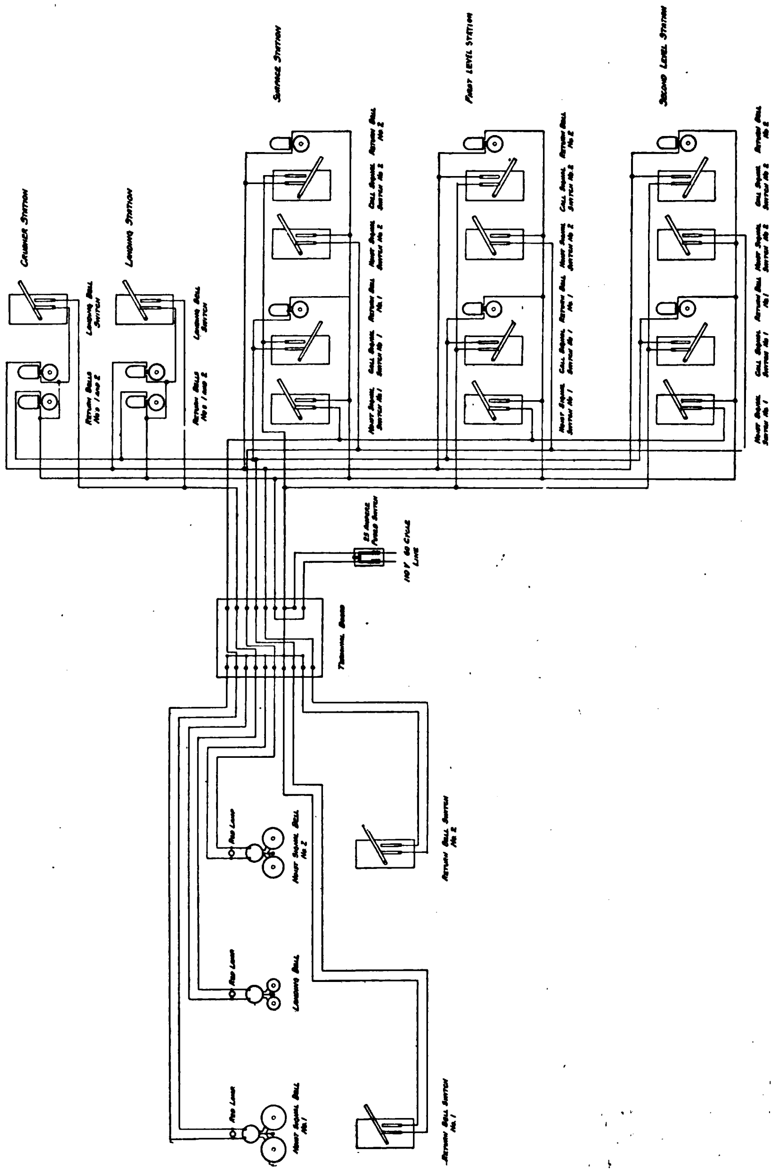


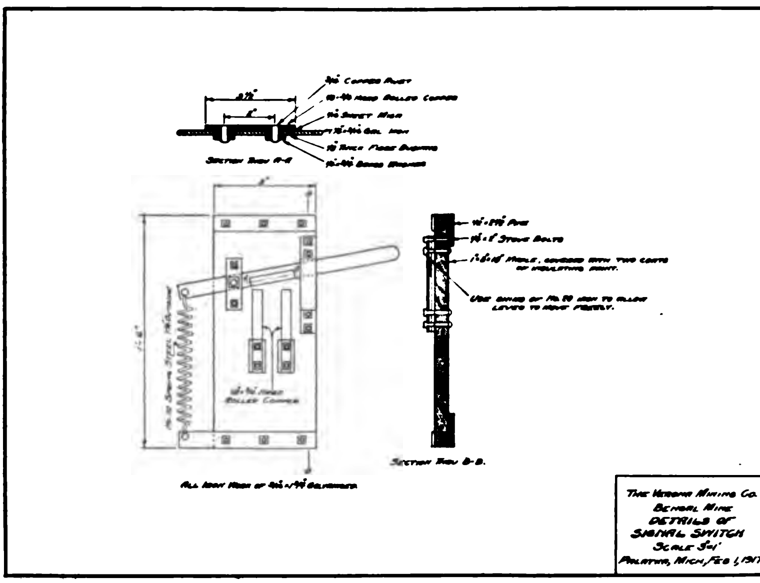
DIAGRAM OF WIRING CONNECTIONS FOR SIGNAL SYSTEM AT BENGAL NO. 1 SHAFT—VERONA MINING CO., PALATKA, MICH., FEBRUARY 23, 1917 and one at the crusher hopper. The switches at these stations operate a separate bell in the engine house. This bell is used to signal both skips, as the only time it becomes necessary to signal from these points is when the chutes or crusher become blocked, or when there is a change in the grade of the material hoisted. Return bells for both skips are placed at these stations, so that the men working in the headframe can tell at all times what work is being done below, and what kind of material is to be hoisted.

The following illustrates the working of this system. A pipeman at second level comes to the shaft to go to surface for fittings and finds that the cage is not in sight. He gives the signal "Send cage to second level" on the call switch, this ringing all the station bells on the cage side throughout the shaft and headframe. A miner and his helper have just come from surface to first level with their machine and tools, and are in the act of unloading these from the cage. The station bell rings the signal, "Send cage to second level." The miner immediately gives the signal, "Cage in use" with his station call switch. The pipeman then knows that he cannot get the cage at once and must wait. In the course of a minute or two, the miner has his tools unloaded, and knowing that the cage is wanted at the second level, signals the engineer on his hoist signal switch, "Move the cage to second level." The engineer repeats this signal with his return switch, and again all of the station bells ring. The pipeman knows that the cage is coming and prepares to continue his trip to surface. When the cage arrives he attempts to give the signal "Hoist to surface," but he makes a mistake in counting and gives the signal, "Hoist very slowly until stop signal is given." The engineer repeats "hoist very slowly until stop signal is given." The pipeman hears the signal correctly and immediately gives signal "Stop," and the engineer answers "Stop." The pipeman then rings again, "Hoist to surface," this time using more care. The engineer answers, "Hoist to surface" and the pipeman boards the cage and makes his trip, leaving the cage at the collar of shaft while he goes to the stock room for fittings. In the meantime a car of rock has arrived at the second level, and the dumpman, seeing neither cage or skip, signals with his call switch, "Send skip to second level loading chute" and receives no answer; he repeats twice, and still no answer. The dumpman is assured that no one is using either skip or cage, but to be doubly sure, he calls on cage side, "Send cage to dump," with the same result. He then signals engineer, "Move skip to second level loading pocket," gets his answer and the skip comes down. It is then loaded and the signal "Hoist rock" is given and answered.

As only ore has been loaded this day, a chute must be emptied and the waste car put in place before the rock can be taken care of in the headframe, and a stop of a few minutes must be made before pulling into dump. As the engineer answers the signal, "Hoist rock" the bells in the headframe station tell the lander and crusher-tender that preparations for taking care of rock must be made, and accordingly one of them signals "Stop" shortly after the skip begins to move. The engineer hears and answers the signal but he does not stop immediately, as he knows that the signal came from the headframe, because it was the landing bell that rang, and that a stop is wanted by the lander before the skip enters the dump. In order to be ready to dump as soon as possible after getting the signal, he pulls his skip within a few feet of the dump before stopping. By this time the pipeman has returned with his fittings and finds the cage gone. He calls several times but receives no answer and then gives the signal, "Move cage to surface," to engineer. The engineer knows that the skip was stopped from the landing and should not be moved until a signal is received from that point. Accordingly, he answers, "Cage in use" to the pipeman and waits for a signal from the proper point. In due time the skip is dumped and the cage sent to surface The pipeman then goes down and continues his work.

One of the most important parts of the whole system is the switch; especially at the underground stations is this true. This switch must be absolutely positive in action, rugged and strong enough to stand the hard usage that it gets, able to stand up and operate successfully when wet and dirty, and easy and convenient to operate. The writer has never been able to get such a switch on the market; so accordingly one has been designed and built on the job. It seems to answer all of the above requirements and even more. The attached drawing shows very clearly the details of the construction of this switch. In general, it consists of a base of hardwood board, heavily oiled and thoroughly coated with insulating paint, upon which is mounted and pivoted at one side an 18-in.

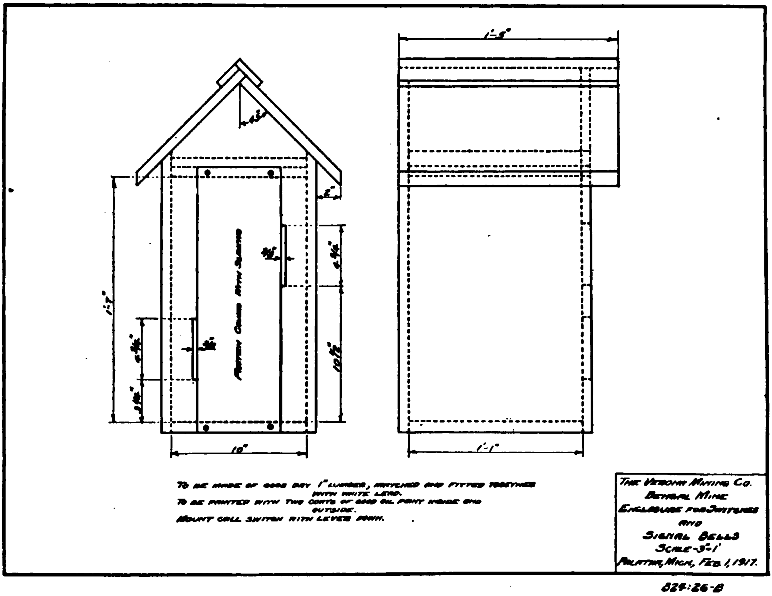
iron lever, held up by a stiff steel spring of No. 10 wire. Upon this lever, but well insulated from it, is a strap of hard copper which wipes upon two fingers mounted rigidly on the base. These fingers are straps of hard-rolled copper or brass 6½ in. long and set ⅛ in. lower than the strap on the lever with which they engage when the handle is pulled down. The spring in the metal is strong enough to hold the surfaces tightly together, and curved tips prevent any danger of bunting on the points. Guide straps and stops are placed in such a way that the lever cannot have any side play and the travel is limited to a fixed



DETAILS OF CONSTRUCTION OF SIGNAL SWITCH—BENGAL NO. 1 SHAFT, VERONA MINING CO.

distance. The whole structure is made very strong and secure, the base being reinforced by 1¼- by 3/16-in. iron straps on top and bottom, the bottom strap acting as a post to hold the tension of the spring. The switch is put together with bolts, so that it is easy to dissemble whenever worn or broken parts are to be replaced.

One of the best features of this switch is the method of making contact and the length of time that the fingers wipe the contact plate at each stroke. The travel of the handle measured on an arc of a circle at the center of the hand grip is $5\frac{1}{2}$ inches. When the hand has traveled one inch the fingers have made contact, and they remain in this position until the lever returns to the same point on the back stroke. The period that the bell is in circuit is equal to the time that it takes the hand to travel 9 in., including a reverse in direction. Observation has shown that the average man when signalling, will make about sixty complete strokes per minute. The pause at the top of the throw is probably a little longer than at the bottom on account of the extra grip that must be taken to overcome the resistance of the spring, while on the bot-



Enclosure for Switches and Signal Bells—Bengal No. 1 Shaft, Verona Mining Co.

tom the tension causes the handle to instantly reverse and aid in pushing the hand up. At any rate, it is safe to say that the time of contact is at least a half a second, which is ample to get a clear, plain signal. It is rather difficult to operate the handle at a much faster rate than this and consequently all attempting to signal too rapidly, very common with the disc and push button type of switches, is largely obviated. The long wipe of over two inches that the fingers make with the contact plate under considerable pressure thoroughly scours off any dirt or corroded metal that may lodge there, thus insuring a perfect contact at all times, even though the switch is in a damp place and only used at infrequent times.

The signal switches and bell are enclosed in a wooden housing, a drawing of which is shown. These boxes are set up securely in convenient places adjoining the compartments they serve. The switches are set in on the sides and mounted so that about 6 in. of the handle protrudes through the slot at the side of the door. Four $\frac{1}{4}$-in. carriage bolts are used to fasten the switch to its housing.

To reduce the chances of making the mistake of using the hoist signal switch when the call switch is wanted, the call switches are mounted inverted, so that the handle must be pulled up instead of down, as is the case with the hoist signal switches. The bell is placed between the switches at the back of the box. To keep unauthorized persons from tampering with the switches and bells, the door is fastened in place with four wood screws. Thus it cannot be taken off very easily without a screw driver, a tool which is seldom found in a mine, but is quickly accessible to the electrician who makes inspections and repairs. The housing, when the door is in place, makes a good serviceable protection to the switches and bell, keeping out dirt and water, as well as tampering fingers. The only opening is the narrow slot in which the handle travels.

The wiring is the most important part of the whole installation, and without doubt is the most difficult to maintain in a satisfactory working order. This was given very careful consideration, and various schemes were examined. The wiring in the shaft furnished the real problem. For this there were three good methods that could be used, namely, armoured cable, rigid iron conduit, and open work. The first two have the advantage that they protect the conductors from injury, and in the case of the armoured cable, from moisture; but both have the disadvantage of being inaccessible for inspection and repairs. This is especially true with the armoured cable. There are several makes of this material that are claimed to be practically indestructible, but the writer has learned by experience that this is not always true under mining conditions. The last method has the advantage that the wiring is exposed to careless workmen's tools and all sorts of falling material, as well as deterioration due to moisture and atmospheric conditions. On the other hand, it has the great advantage of being quickly and cheaply repaired in case of accident. The enclosed ladder compartment was in itself a sort of a conduit and its contents were fairly well protected from falling material. It was decided that if larger wires and better insulation were used than the current and voltage required, the open work method would be the most satisfactory. The adjacent ladders would make every inch of the line easily and quickly reached at any time. Accordingly No. 10 Okonite double-braid solid wire with 3/32-in. rubber wall was used, as this is the best material known for this class of work. The six wires were suspended by strain insulators just below the collar of the shaft and dropped through porcelain tubes held in racks spaced every 10 ft. all the way down. At every second rack a band of friction tape was wound tightly around each wire at the top of the tube until the diameter of the band was considerably larger than the hole in the tube. This helped to support the weight of the wire, and if a break occurred, saved the broken end from slipping down the shaft. No taps were made in the shaft, but at the stations two short pieces of 1¼-in. conduit were run from the switch box into the ladder compartment and the six wires looped in and out through the conduits, the switch box being used as a terminal and junction box.

Between the shaft and the engine house the signal wiring was run through a multi-duct clay conduit laid underground and carrying all power and lighting circuits. The signal cable through the duct is a seven-conductor lead-covered standard Okonite 600-volt underground cable, the conductors being of No. 12 wire. This cable after passing through the duct terminates in a steel cabinet in the engine house. In this cabinet a slate terminal board is mounted, upon which are binding posts for every wire of the system at the engine house end. Each post is stamped with letters and figures to show exactly which wire terminates there. All necessary cross-connections are made on the back of the slate. The board is designed so that in case of trouble any circuit can be disconnected and tested out, or any part of the system can be cut out and the remainder kept in use.

The switches and bells at the engineer's station are mounted on a pipe framework set into the concrete of the hoist foundation. Each unit is served by No. 14 N. E. code wire drawn through a ½-in. conduit laid in the concrete floor of the building and terminating in the panel-board cabinet.

The signal bells are a very important part of the installation, but since there are a number of makes on the market that operate very satisfactorily on 60-cycle 110-volt circuits, the problem was easily solved. For the outdoor bells, including the underground stations, a 5-in. weather-proof vibrator was chosen. This bell has laminated core magnets with impregnated coils. The make-and-break points are of carbon, easily adjustable by a setscrew on the outside of the case, and can be quickly renewed at a trifling cost when worn away. These bells give sharp, clear-cut, and vigorous signals and require very little attention. In the engine room the bells used were of the polarized type, with double gongs. Bells of different sizes are used, to give variation in tones. The three gongs are 10, 7 and 3 in. in size, the 3-in. gong giving the landing signal. In addition to the bells, 30-watt red carbon lamps were connected across the terminals, to flash at each stroke of the signal switch. This gives a visible as well as an audible signal, and is very useful, especially when the hoist is running and there is considerable noise.

A spare bell and a spare switch are kept on hand at all times, but it is very seldom that it ever becomes necessary to change a bell, and a set of carbon points last about one year. The fingers on the switches that get the greatest amount of work need to be renewed about once every fifteen months.

This system was installed in September of 1913, and has never caused a moment's delay in hoisting up to present time. The apparatus is in excellent condition and from all appearances will stand for many more years.

STOPING TO BRANCHED RAISES.

BY F. W. SPERR.

Branched raises are used for tapping the underside of orebodies or blocks of ore, of considerable size, to convey the broken ore from above to more or less centralized loading chutes on the haulage level. They have been used for sub-level stoping and block caving; and might be (and possibly have been) used for milling and for chute caving. To illustrate their use for sub-level stoping let us assume an irregular lenticular orebody. Fig. 1 represents the horizontal outlines of such an orebody at 25-ft. intervals. The wall-rock and capping are strong, and the ore is of medium hardness and texture. Therefore large chambers of ore may be extracted without caving or in any way disturbing the enclosing rock. The main considerations in applying a stoping method are: safety to the workmen, the elimination of timber, and the reduction of shoveling and tramming. It appears that the ore may be extracted from the top downward, leaving the opening self-supporting, thus eliminating the use of timber. The reduction of shoveling and tramming may be accomplished by a suitable arrangement of "mills" into which the ore falls as broken, and from which it may flow directly into tram-cars on the main haulage levels, under control by suitable chutes.

The first haulage-way will be on the foot-wall 100 ft. below the top of the ore. Four vertical raises will be driven on the foot-wall side, 45 ft. apart and designed to be near the hanging wall at the top sub-level, which in this case is 75 ft. above the haulage level. If no other raises were used, a great deal of shoveling and tramming would be necessary in the stopes; therefore, branched raises are started from the main vertical raise in such a way as to cut up the block of ore most advantageously for reducing the shoveling and tramming. For maximum efficiency the raises should be 25 ft. apart on every sub-level, should reach close to the foot-wall, and should approach the hanging wall according to its dip; the steeper the dip the closer the approach.

Beginning at the top sub and working downward, drifts and cross-cuts are made 5 or 6 ft. wide and 6 or 7 ft. high on the different sub-levels to connect the raises from around the sides, and by means of down-holes the tops of raises are made funnel-shaped and widened, until their rims become contiguous. This leaves 15 to 17 ft. of ore in place above the cut. By means of upper holes, about 8 ft. of the overhanging ore is brought down and left lying for a foundation upon which the drills are rigged for another series of upper holes. If, in places, the ore goes up more than 25 ft. above the sub, as it often does in irregular fingers, it is mined by successive series of upper holes, or by the ordinary back-stoping-on-broken-ore method.

The sub-levels are 25 ft. apart vertically. Each sub is mined in substantially the same way as the top sub, except that the "mills" will be better distributed on some sub-levels than on others. There will always be the balancing between the cost of extra raises and the cost of extra shoveling and tramming.

Fig. 2 illustrates any one of the different vertical sections on the lines AA, BB, and CC, of Fig. 1. The section on the line DD would show the bottom of the raise in the foot-wall rock.

Fig. 3 represents the vertical sections through the raises B and C, on the lines BF and CF, at right angles to the lines AA, BB, etc. The branches are driven in four directions from these main raises, in two sets one above the other, and at vertical angles of about 55 degrees. The branches from A and D are driven in three directions, the fourth direction being against nearly vertical walls. By this arrangement mills are provided on the 75-ft. sub-level, and distributed in such a manner that little shoveling and tramming will be necessary. Fig. 4 shows the positions of the raises on this sub and the connecting drifts and cross-cuts. The raise D2 to the right is extended on the pitch of the formation.

Fig. 5 shows the positions of the raises on the 50-ft. sub, connected by drifts and cross-cuts. The arrangement of the mills at this level is nearly ideal, as illustrated by Fig. 6, which shows the approximate positions of the rims of the mills after being funneled out. Hog-backs of ore will be left lying on

MINING

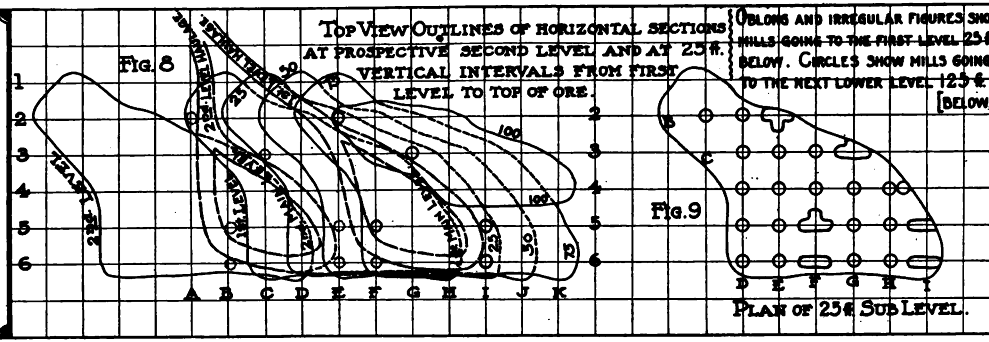
LENTICULAR ORE BODIES

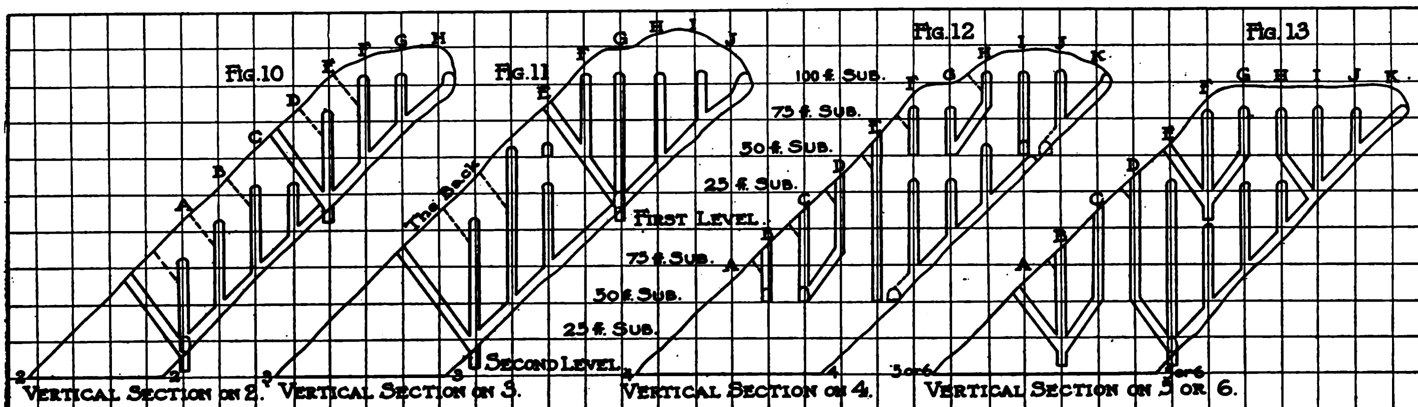
SUBLEVEL STOPING TO BRANCHED RAISES

— Scale —

IRON ORE MINING THROUGH BRANCHED RAISES

F.W.S.





the spaces standing between the mills. All this goes down with the next lower sub. But along the foot-wall, all broken ore has to be cleaned off to the sub-floor. Otherwise it is liable to "hang-up" instead of coming down with the next lower sub; therefore it is desirable to have the raises along the foot-wall close enough together and close enough to the wall to prevent any considerable quantity of ore from finding lodgment.

The branched raises are available for mills on the higher subs, but not on the first sub above the main level. Fig. 7 shows all that remains of the raises on the 25-ft. sub-level. Only the main vertical raises are left; but these are well distributed along the foot-wall, making the amount of shoveling and tramming the least possible under the circumstances. If the ore does not extend below the main level, immediately under this sub, it will probably be more economical to do the shoveling and tramming, than to provide more mills together with the necessary chutes. But, if the ore extends downward so that another main level becomes necessary, some of the ore from the 25-ft. sub may best go down to this next lower level through some of the mills, which may be extended upward as required. Ordinarily the ore from each main level slice is milled to the chutes of the next lower level.

Various names have been given to this method of mining. It is variously known as "subbing," "underground milling," "slicing without timber" and "sub-slicing without timber." The last term seems the most distinctive. It suggests the method of operating and does not apply to other methods, while all of the other terms are used also to designate methods that are much different in detail; but it would seem better to cut out the term sub-slicing altogether; and in this case say sub-level stoping. The term subbing generally means sub-level stoping; but sometimes it means sub-level caving which has also been called sub-slicing.

Fig. 8 to 18 illustrate the case of an orebody beginning somewhat irregularly under the capping and extending downward with a nearly vertical hanging wall, and with the "back" pitching down at an angle of 45 degrees. The physical conditions are similar to those in the preceding case, and the same method of mining will be applied with little difference in detail.

Branched raises from six chutes on the first level and

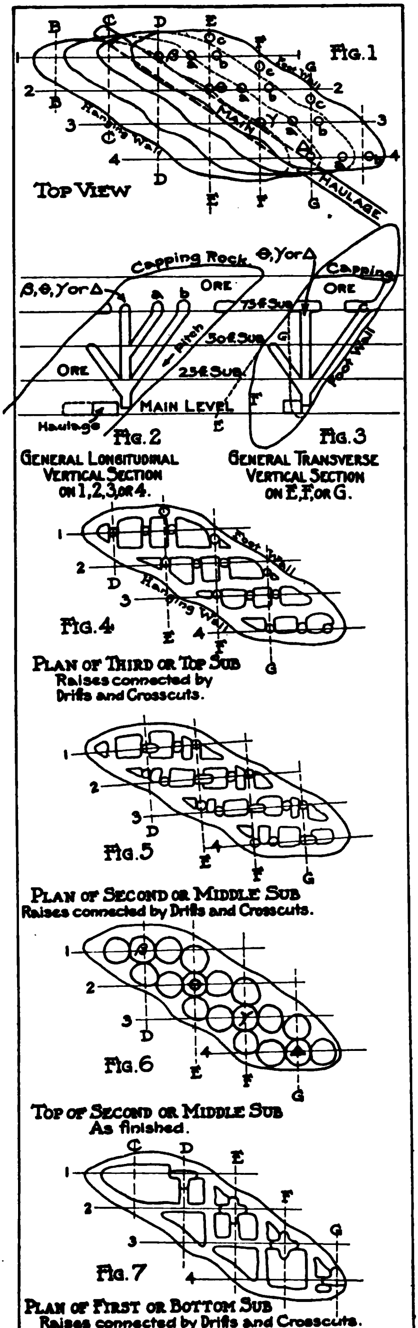


Fig. 2 General Longitudinal Vertical Section on 1, 2, 3, or 4.

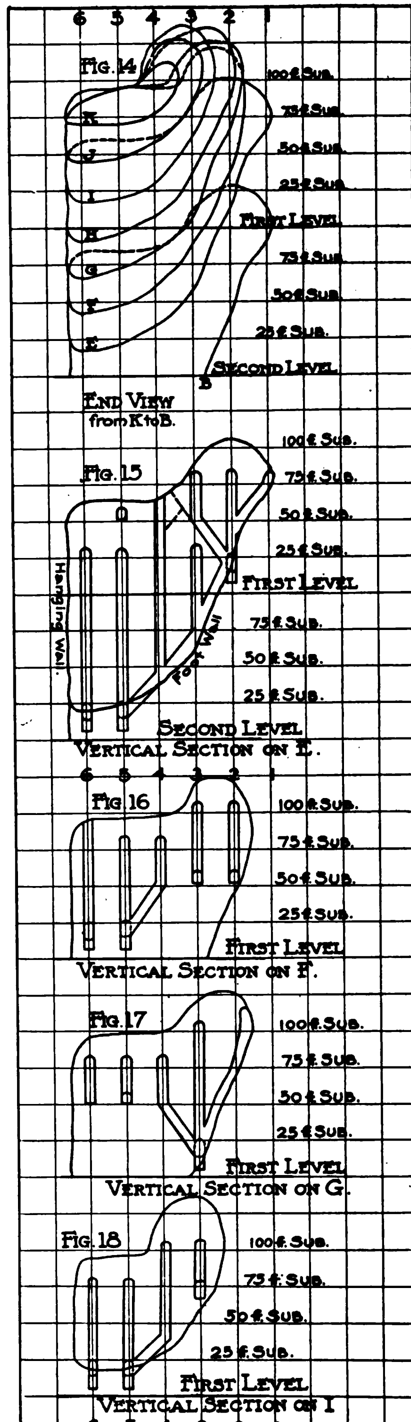
Fig. 3 General Transverse Vertical Section on E, F, or G.

Plan of Third or Top Sub. Raises connected by Drifts and Crosscuts.

Plan of Second or Middle Sub. Raises connected by Drifts and Crosscuts.

Top of Second or Middle Sub. As finished.

Plan of First or Bottom Sub. Raises connected by Drifts and Crosscuts.



End View from K to B.

Vertical Section on E.

Vertical Section on F.

Vertical Section on G.

Vertical Section on I.

from the same number on the second level, provide mills 25 ft. apart on every sub-level. The foot-wall sides are properly provided, and nearly all shoveling and tramming in the stopes is eliminated.

The ore is of such a character that it gives little trouble by packing in vertical raises; therefore the branches are made vertical in places for convenience of arrangement. Accessibility to the vertical mills for the purpose of relieving the tendency to pack, can be more readily provided where the branches are inclined throughout their entire length, than in the case where they are partly vertical. Softer ore has a greater tendency to pack in the mills at a high angle, as well as to hang up in the mills at lower angles. The best angle for the worst ores, in this regard, is 65 degrees, above horizontal, but some ores run freely at an angle of 45 degrees.

The developments from the second level are as yet only prospective; but it is reasonable to suppose that the ore will go down another 100 ft. judging from the developments of the first level. The second level should be developed no earlier than necessary for continuous production to follow the exhaustion of the first level; for by long standing the chutes decay and the openings cave, and the whole foundation may be weakened to such an extent as to destroy the system of mining. Fig. 8 is a top view of the outlines of the horizontal sections of the orebody at the prospective second level and at vertical intervals of 25 ft. from the first level to the top of the ore. The full circles show the positions of the chutes on the first level. The dotted circles show the positions that the chutes will occupy on the second level, if there is no material change in the formation in the next 100 ft. downward. Fig. 10, 11, 12, and 13, represent vertical sections on the lines 2, 3, 4, and 5, respectively. The vertical section on the line 6 is substantially the same as the one on the line 5. These sections, together with the transverse vertical sections on the lines E, F, G, and I, as shown in Fig. 15, 16, 17, and 18, show the complete system of development of the raises for the first and second levels.

The plan of any one of the sub-levels below the irregular top, as developed by raises from two successive main levels, will be typical of all. Such a plan is represented by Fig. 9. The circles are the tops of the mills going to the second level.

The oblong and irregular figures are the tops of the mills going to the first level. Plans of different sub-levels will differ from each other in little except the relative positions of the two different classes of mills.

THE ORE MINING METHOD USED AT THE RAIMUND DIVISION, BIRMINGHAM DISTRICT, ALABAMA.

BY GERALD G. DOBBS.*

OUTLINE:

1. Location of the property.
2. General description of the property.
3. Description of the orebody.
4. Methods of mining.
   (A) Method used in mining the orebody above the fault.
      (a) Soft-ore mining (outcrop workings and scrams).
      (b) Hard-ore mining (slope sinking, driving and robbing).
   (B) Method used in mining the orebody below the fault.

1. LOCATION OF THE PROPERTY:

The Raimund Mining Division of the Republic Iron & Steel Company is located in the Birmingham district of Alabama, about fourteen miles southwest of the City of Birmingham, and two miles south of Bessemer. The Birmingham District is centrally located in the State of Alabama, and in the new industrial South, and takes its name from the progressive city of 180,000 population about which the industries of the district are closely grouped. The phenomenal growth and development of the district is due to the circumstances that nowhere else in the world are the raw materials necessary for the production of pig iron—iron ore, dolomite, limestone and coking coal—located in such close proximity.

At present the principal iron mines of the district are located on Red Mountain, and extend in a row at intervals of about 2,000 feet, for a distance of fourteen miles from Birm-

*Superintendent Raimund Mining Division, Republic Iron & Steel Co.

ingham to a point below Bessemer. The Raimund Mining Division is situated on the extreme southwest end of the active mining area. Red Mountain, a long ridge with a general trend N. 30° E., is in reality the side of an anticline, the apex of which was originally over Jones Valley, and which has been eroded to a depth of 300 feet below the crest of Red Mountain. The outcrop of the "Big Seam," the hematite seam, which at present is being worked by all the important mines of the district, extends in a practically unbroken line at or near the crest of Red Mountain from Birmingham to a point about four miles below Bessemer.

2. General Description of the Property:

The Raimund Mining Division has an area of approximately 1,200 acres, including about 1½ miles of the "Big

Change House, Raimund

Seam" outcrop and an ore-bearing territory extending to a depth of about 1½ miles from the outcrop.

The ore at present is being mined from three slopes sunk on the dip of the ore seam. Also some outcrop "soft ore" is being mined and loaded directly into the railroad cars through a "ramp." On the slopes, 10-ton steel skips of 5-ft. gauge are used. These dump automatically into either ore or poor-rock pockets. The ore passes to gyratory crushers and thence directly to the railroad cars, and the poor rock is trammed to the rock dump. The skip is sent to the right pocket by throwing in place the proper section of the dump rail. This is controlled by a piston operating under compressed air.

The hoist engines used at Raimund are 30 by 60-in. Nordberg, first-motion hoists with drums 10 ft. in diameter grooved to take a 1⅜-in. rope. They are equipped with Johnson Hoist Recorders, which supply accurate information as to the loading and hoisting work done. The air compressors are Nordberg, Allison, and Ingersoll-Rand. The boiler plants are equipped with 72-in. by 18-ft. return-tubular boilers operated in connection with Cochrane and Blake-Knowles feed-water heaters.

The Raimund camp contains 203 houses for employes, two

PUBLIC AND DOMESTIC SCIENCE SCHOOLS FOR NEGRO EMPLOYEES—RAIMUND

commissaries, a brick change house with 525 steel lockers and 22 showers, a public school and a domestic science school for colored employes.

3. DESCRIPTION OF THE OREBODY:

The Raimund orebody is a fossiliferous hematite seam outcropping on the side of Red Mountain and dipping in a southeasterly direction under Shades Valley. The portions of the seam nearest to the outcrop show dips varying from 20 to 45 degrees. Within the last few years, however, all of the Raimund slopes have encountered a fault with a general northeast trend; however, as this fault progresses northeastward, it diverges from the outcrop. Exploration by diamond drilling has disclosed a maximum downward displacement of the orebody of 450 feet, the fault having a hade of about 80° in the direction of the displacement. The portion of the orebody below the fault and adjacent to it has since been found to have a maximum dip of 25°, this dip decreasing in a southeasterly direction until in some places the ore seam is horizontal and even rolls so that portions of the seam dip to the northwest.

The upper portion of the "Big Seam," extending from the outcrop to a distance of approximately 350 ft. along the dip, has been leached by percolating water, with the result that

No. 1 Mine—Raimund

the lime carbonate has been largely removed. As the removal of one constituent increases the relative percentages of the remaining less soluble ingredients, this remaining "soft ore" is higher than the rest of the ore in iron and silica. A typical analysis of Raimund soft ore would be as follows:

Iron.	Sil.	Alum.	Lime.	Phos.	Man.
47.52	20.98	5.41	1.39	.39	.26

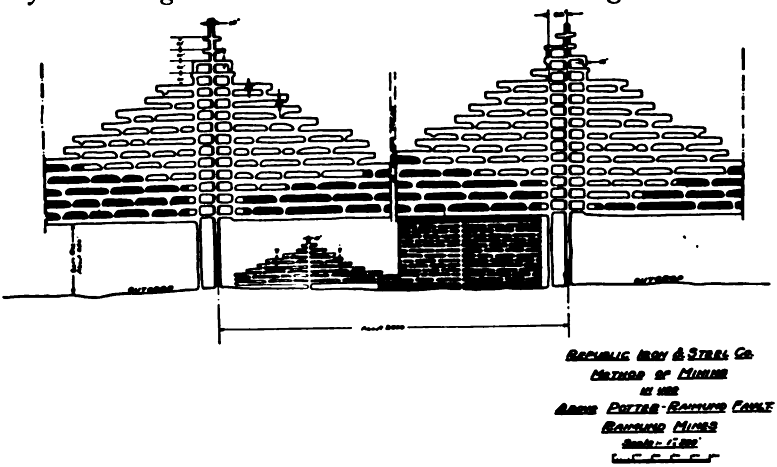
The lower portion of the seam, on which the major portion of the mine workings are located, is called "hard ore," because of its structure, and because it still carries much lime, not having been leached. A typical analysis of this iron, which is practically self-fluxing, would be as follows:

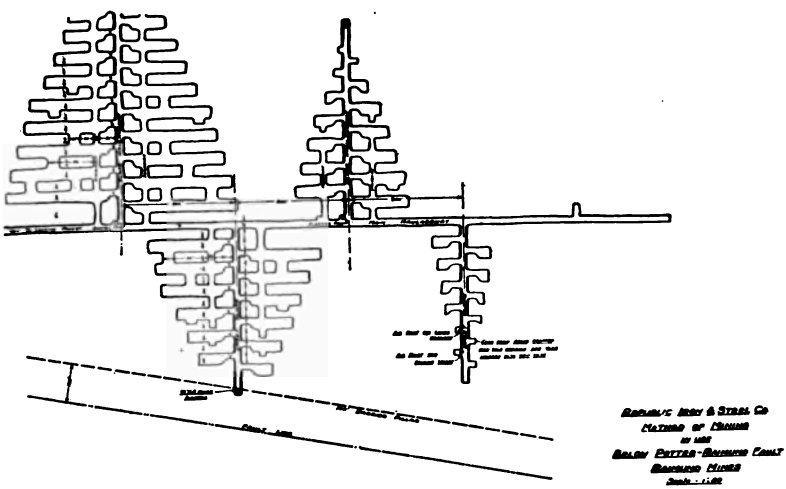
Iron.	Sil.	Alum.	Lime.	Phos.	Man.
34.04	12.46	3.88	18.13	.29	.19

4. METHOD OF MINING:

(*A*) *Method Used in Mining Orebody Above the Fault.*

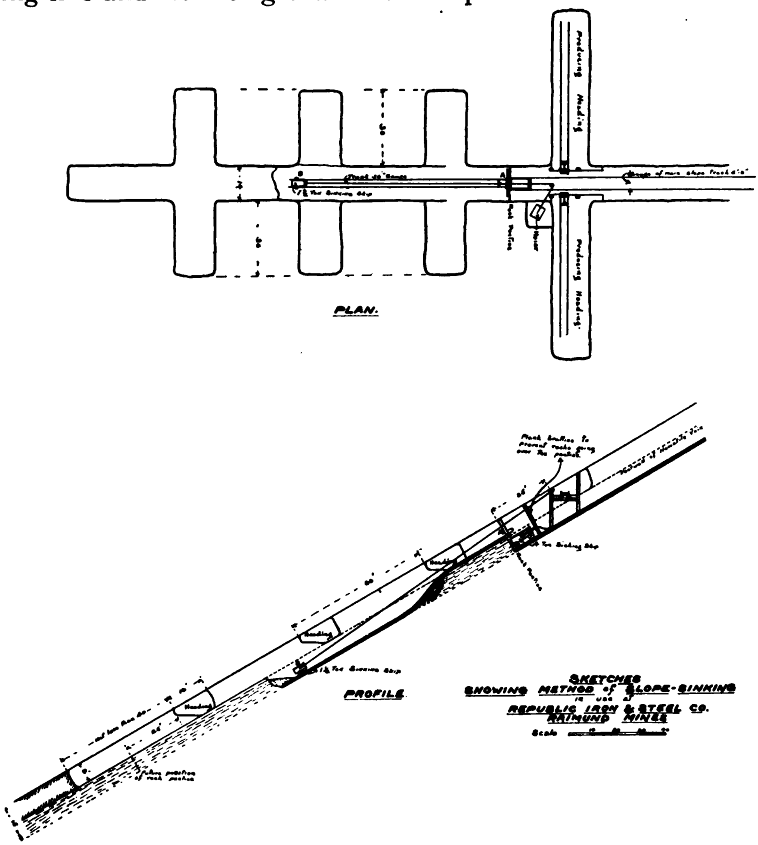
(a) Soft-Ore Mining—The outcrop ore is mined by removing the overburden and then mining the ore in





benches. This method is possible only up to a point where the overburden reaches a thickness of about 12 feet. Some outcrop ore is being mined at the present time and is being transferred by wagons to a "ramp," where it is dumped through a trap door directly into the railroad cars.

The blocks of soft ore lying between the main hard-ore slopes, are being mined from temporary timbered slopes, known as "'scrams," sunk on the ore seam. The "scram" slopes are usually driven with an 8- by 10-ft. cross-section and headings are turned at intervals of 25 feet. These headings are then driven 15 ft. wide to the driving line. The remaining 10 ft. pillar is then robbed, the robbing commencing at the driving line and retreating toward the slope.



(*b*) *Hard-Ore Mining.*

Slope Sinking—As before stated, the zone of soft or leached ore extends as an irregular band roughly parallel to the outcrop at a width of about 350 feet. This soft ore zone, as can readily be understood, is rather heavy ground; so it was decided to drive well-timbered slopes completely through it before starting mining proper, and also to confine the mining operations of these slopes to hard ore only. The timbered upper portions of these slopes have since been replaced with reinforced concrete.

The soft ore lying between the main slopes, which are approximately 2,000 ft. apart, was to be mined later by scrams. When this was done, 40-ft. ore pillars were left between the nearest scram workings and the side of the manway, and 75-ft. pillars between the nearest scram working and the side of the slope on the opposite side from the manway.

In opening up the hard ore, manways are driven parallel to each slope, a 75-ft. pillar being left between each manway and slope. Then slopes 14 ft. in width and 8 by 44 ft. in cross-section, 8 ft. being the width of the ore seam on this

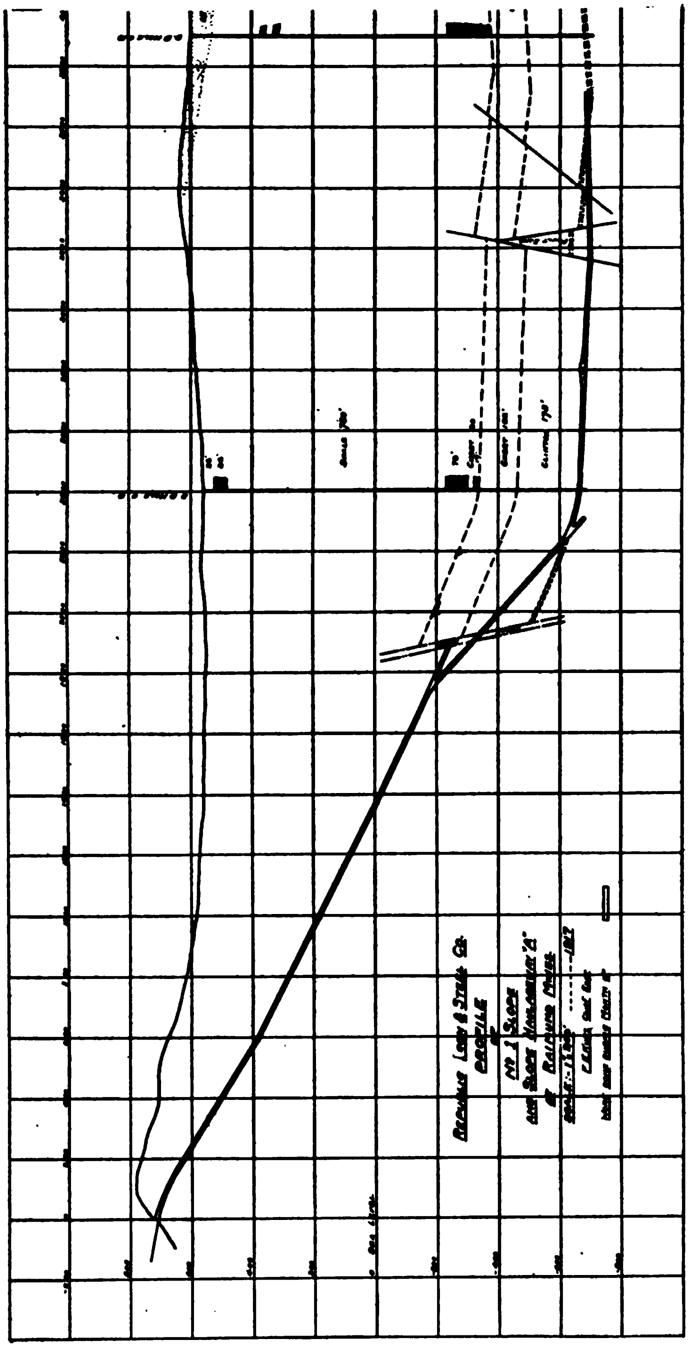
OFFICE AND COMMISSARY—RAIMUND

property, are sunk in the ore. The footwall is then taken up for a depth of 6½ ft. to allow head room later on for the dumping of heading tram cars into the 10-ton steel skip. In this way, a slope is developed approximately 14½ by 14 ft. in cross-section. Headings 15 ft. wide are turned at 60-ft. intervals along the dip, being driven from the slope along the ore seam. The ore slope is usually driven about 70 ft. below the lower rib or side of the lowest heading, before the footwall is taken up, and the footwall is then taken up only to a distance of about 25 ft. below the lower rib of the lowest heading. The slope sinking in detail for one cycle can be understood from the accompanying sketch. Here the ore is 70 ft. ahead of the heading rib and the end of the footwall is cut 25 ft. ahead. Stringers are laid along the footwall and a horn tipple is placed at A. At the same time a hoist engine is placed in a 10 by 10-ft. engine room cut in the ore at one side of the slope just below the heading above the pentice. The sinking skip B is a 1½-ton skip, and the track is 36-in. gauge.

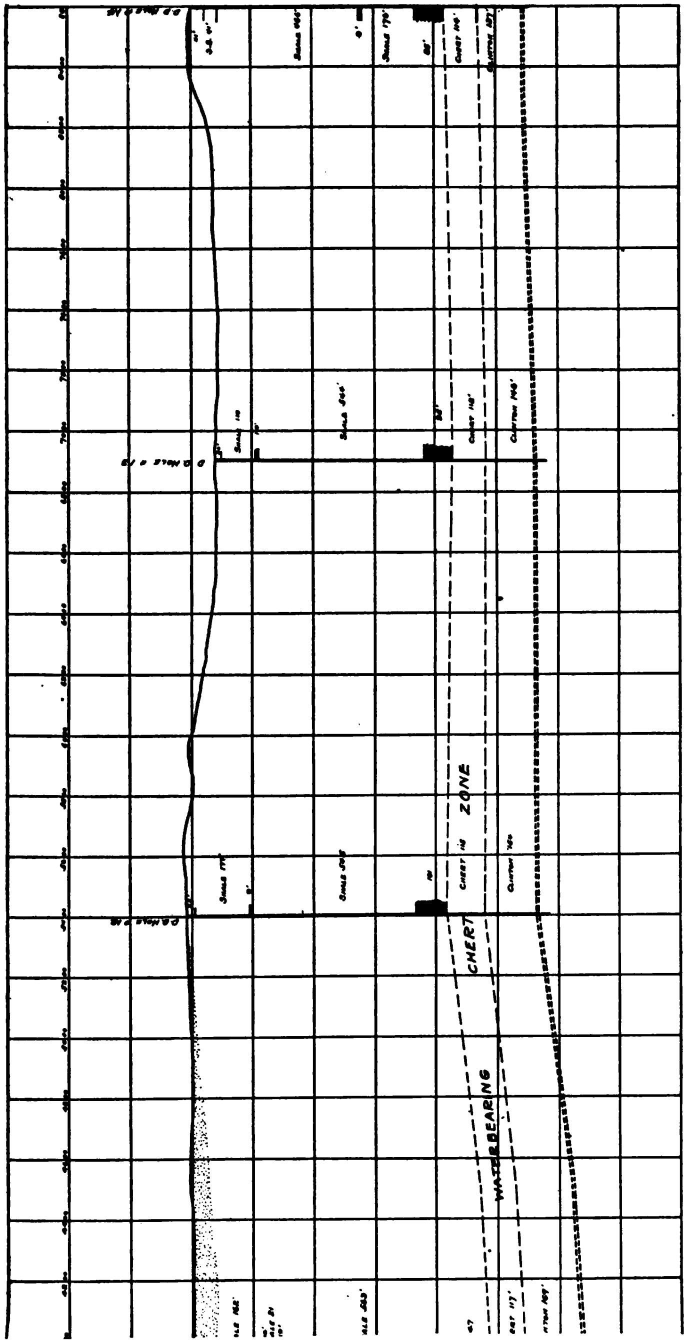
The slope is then driven ahead in the ore. Whenever the face reaches 70 ft. below the lower rib of the heading above, two new headings are turned, one on each side of the slope and 60 ft. below the nearest headings as measured along the dip. These headings are driven in about 30 ft. from the side of the slope. In this way, as these two headings are being

SUPPLY HOUSE AND SHOPS—RAIMUND

driven in from the slope, a section of the slope 10 ft. in length is serving as a sump. As soon as the headings are driven in the required distance, slope sinking is resumed, and continued until three pairs of headings have been turned and an additional 70 ft. of slope below the lowest heading has been driven. The track is then pulled up, the footwall blasted, at the pentice first, and the track laid on this steepened pitch to a point 6½ ft. below the ore. The track is then laid, being placed parallel to the footwall of the ore seam but 6½ ft. below it. The full 6½ ft. depth of footwall is then taken up as the work progresses to a point, as before stated, 25 ft. below the lower rib of the lowest heading. The pentice tipple



SECTION 1



SECTION 2

is then removed, the pentice shot up, and the main slope track continued on down to the face of the footwall cut, where the next pentice will be placed. Heading tipples and tram cars are then placed in the six available headings and mining is begun, the tram cars being dumped into the big skip in which the ore is hoisted to surface. At the same time the pentice tipple is put in position and the slope-sinking cycle is resumed.

Driving—After the tipples have been put in place the headings are driven 15 ft. wide for a distance of 102 ft. from the center line of the slope. Manways are then driven up the dip parallel to the slope until they break through into the headings above. These manways are started 20 ft. wide, but

"SOFT ORE" RAMP IN FOREGROUND—NO. 2 TIPPLE IN BACKGROUND—RAIMUND

decrease in width until breaking into the heading above, when the hole is rarely over 8 to 10 ft. wide. After passing the manway, the heading is widened to about 30 ft. and driven to the "driving line," an arbitrary line dividing the territory to be mined from the slopes on either side. This driving line, when two slopes are on the same property, is a line bisecting the angle of intersection of the center lines of the two adjacent slopes. Whenever the natural ventilation becomes defective or insufficient, "upsets" or raises are driven through the pillar to the heading above. As the headings are turned at intervals of 60 ft., measured along the dip, with a heading width of 30 ft. after subtracting the ore which is mined while the upsets are being driven, the ore left in the pillars at commencement of robbing usually amounts to about 40 per cent. of that originally blocked out for removal by any particular heading.

Robbing—When a heading reaches the driving line, if the robbing in the heading above has progressed a sufficient distance towards the slope, robbing is started. An upset is first driven along the side of the driving line to the heading above, and then successive slices are shot from the side of the upset nearest to the slope. As the robbing face retreats toward the slope, the hanging wall and overburden are allowed to cave. Rails, spikes and ties, if in good condition, are pulled

Skip Dumping at No. 1 Tipple Showing Arrangement for Dumping Either Ore or Poor Rock—Raimund

up and used in future workings. When a point 40 ft. from the side of the manway is reached, robbing is discontinued and the heading is considered worked out until such time as the property is to be abandoned, when the manway and slope pillars will be robbed and mining operations entirely discontinued.

The drilling is carried on with D-24 Ingersoll-Rand drills at an air pressure of 75 to 80 pounds per square inch. A miner, a drill helper and three muckers constitute a heading crew, such a crew mines and trams 50 tons of ore daily. The ore is loaded into tram cars of 40 cu. ft. capacity, which hold about two tons of broken ore. These cars are trammed by gravity along the heading track, usually driven up at a grade of about 2.5 per cent., to the tipple at the slope, where the ore is hoisted to surface. The empty cars are hauled back to the working face by mules, one mule and muleboy usually serving three headings.

The tram cars are made of oak plank held in an iron framework, the sides being of 2-in. and the bottom of 3-in. plank. The upper edges of the side planks are covered with light angle-irons to protect them against undue wear during loading. Ninety per cent. of the mine labor is negro. The white men are usually pump men, pipemen, development contractors or foremen. Headings usually require little timber;

No. 2 Tipple—Raimund

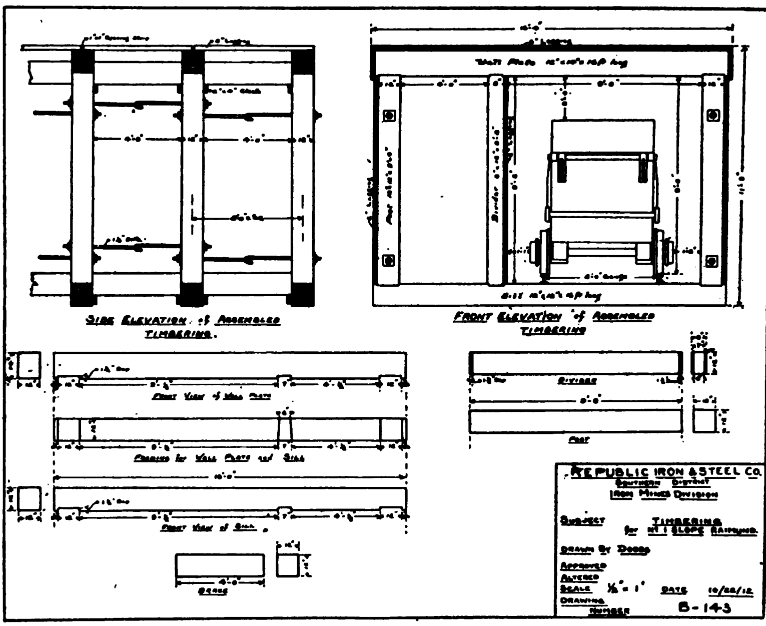
but if needed, long-leaf pine props 10 in. in minimum diameter are used in driving work, and 7 in. in diameter in robbing work. The life of the average framed pine timber cut in this district is from 4 to 5 years, while the unpeeled heading props rarely last over 2½ to 3 years.

(*B*) *Method Used in Mining the Orebody Below the Fault.*

When the main slopes encountered the big fault and the ore was found to be displaced downwards, it was decided to steepen the dip of No. 1 and No. 2 slopes and drive them ahead to intersect the lower orebody as soon as possible. Accordingly two-compartment timbered shafts were sunk, in No. 1 slope at an angle of 45 degrees and in No. 2 slope at an angle of

50 degrees. The timbering used in the two-compartment shafts can be understood by studying the detail drawing of it. The main features to be noticed in this design are that the timbering was designed to withstand a pressure at right angles to the dip, and that, except for the divider, all the framing consists of square cuts and laps.

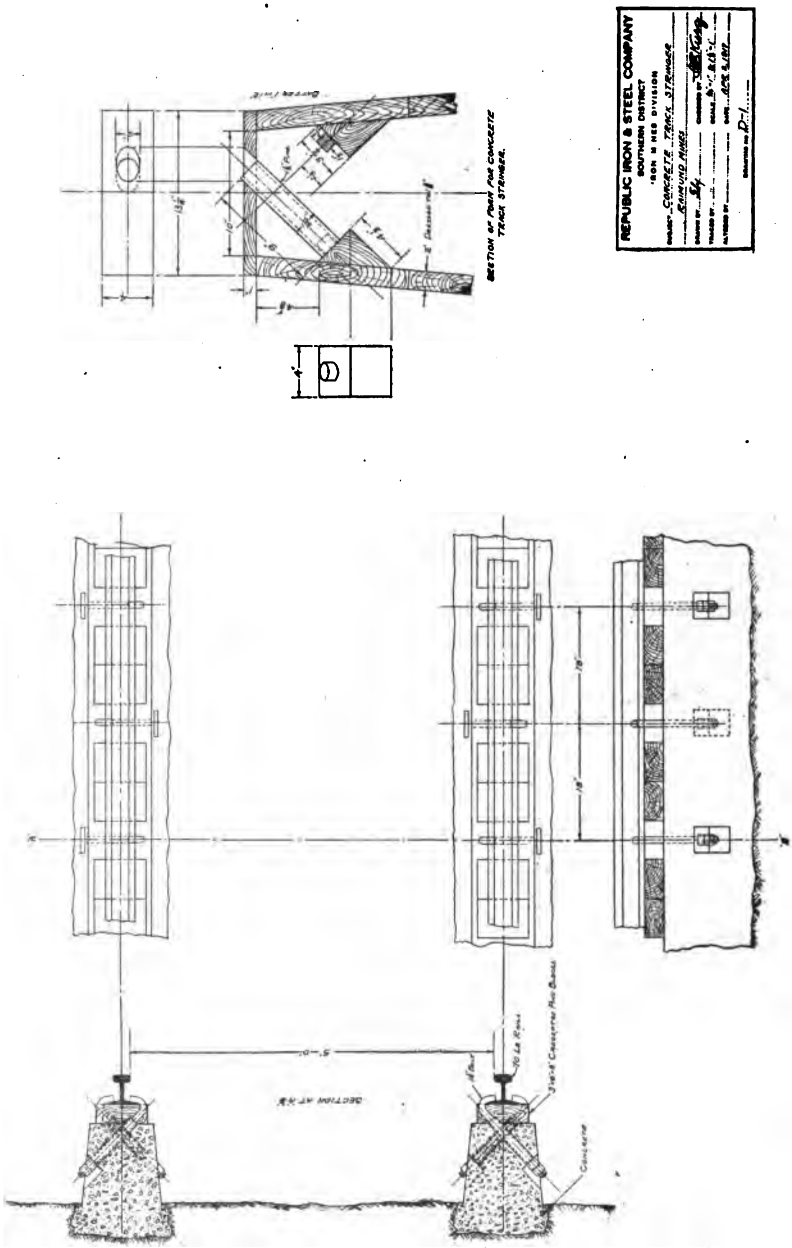
After the inclined slopes were driven to the ore below the fault, the footwall was shot up and the change from the flatter dips of the upper seam to the steeply inclined shafts made by 10-degree vertical curves. As rapid hoisting was desired,



the skip track was placed permanently and with great accuracy. Concrete track stringers were placed under both rails over the curve and for some distance on the tangents at both ends of the curves. The concrete track stringer, a detail drawing of which is shown, was anchored to the footwall, in an 18-in. trench hitch, by many iron pins placed in holes drilled in the bottom of the hitch, about which the concrete was placed when the stringer was poured. This concrete track stringer, although similar to the "Mohawk track stringer," has several features which make it a decided improvement over that design. These differences can be summed up as follows: First, the Raimund design supplies three times the bearing area for the rail on the wood cushion blocks. Second, the Raimund hook bolts are placed on alternate sides of the stringer 18 in. apart, and at the point where they clamp on to the rail they are in an open space and not, as in the Mohawk type, placed in the wood cushion itself. This arrangement makes replacing and gauge adjusting easier.

When the inclined shafts intersected the ore seam, loading and measuring pockets were installed and development was pushed with all possible speed. Since that time, however, due to the encountering of many minor faults and rolls, neither of the slopes has been opened up so that a maximum tonnage can be handled.

At present, the following method of mining is being followed: Main haulage-ways 8 to 15 ft. in cross-section are first driven in the ore to provide for double-track handling of tram cars. From these haulage-ways, upset slopes 12 ft. wide are driven up the inclined footwall at intervals of 600 ft. along the haulage-way, and when the ore dips below the haulage-way, slope haulage-ways are sunk at intervals of 600 ft. measured along the haulage-way. From these upset slopes and slope haulage-ways, headings 15 ft. wide are turned at intervals of 60 ft. measured along the dip and driven in at this width for a distance of 20 ft. when they are widened to 30 ft. and continued at this width to the driving line. At first these headings were turned opposite to each other, but in later workings they were staggered so as to facilitate the placing of knuck tracks. Since the maximum length of any heading in a slope haulage-way or upset slope is about 300 ft., mule haulage is unnecessary and the cars can easily be trammed by the muckers themselves. Manways are driven on both sides of the upset slopes and slope haulage-ways, and are so placed as to leave 40 ft. pillars on each side of the slopes. Upsets or raises 30 ft. wide are driven through the heading pillars at intervals of about every 90 ft. to help the ventilation. Electric hoists with 75 h.p. motors are placed at the upper ends of the upset slopes or slope haulage-ways to transfer the cars from the headings to the main haulage-ways, where they run by gravity to the pocket. The empty cars are returned to the upset slopes and slope haulage-ways by mules. The underground mule stables are ventilated by bleeders on the compressed air lines, the mules remaining underground perma-


REPUBLIC IRON & STEEL COMPANY
SOUTHERN DISTRICT
IRON MINES DIVISION
CONCRETE TRACK STRINGER
RAIMUND MINES
SECTION OF FORM FOR CONCRETE TRACK STRINGER.

nently. The ore in No. 1 slope when placed in the measuring pocket is dumped into the skip in 8-ton loads through a door in the bottom of the pocket. The door is controlled by a toggle-joint mechanism similar to those in use in the Newport mine, Michigan, being operated by a compressed air cylinder. In No. 2 slope, the ore after being placed in the measuring pocket is dumped into the skip by releasing the entire bottom of the pocket, which consists of two doors hinged at the sides of the pocket. When in a horizontal position, these doors are held in place by a spring latch. The latch is released by a hand lever. The doors are swung back into position by a counter-weight, and the measuring pocket is ready to receive another 8-ton load. From the measuring pocket the skip is hoisted to the outside tipple, automatically dumped, and lowered to the measuring pocket. The round trip can be made in about 3½ minutes, which time includes both the time consumed in loading the skip at the measuring pocket and the time required to dump it in the outside tipple.

The system of mining at present followed below the big fault recovers about 60 per cent. of the orebody and leaves approximately 40 per cent. in the pillars to support the hanging wall. The reason for this is that there is a stratum of water-bearing chert about 150 ft. above the ore seam, and should the formation be caved up to this stratum, the pumping of this water would be, if not impossible, at least a severe handicap upon all future mining on the property. Diamond-drill holes which have been drilled from the underground workings and which have cut the chert formation, on being plugged have shown water pressures above 200 pounds per square inch. At some future date when the long haul and the handling become too expensive, a shaft will have to be sunk is Shades Valley. A very difficult problem will then have to be solved, namely, the sinking or raising of a 1,000-ft. vertical shaft which at a depth of 700 to 850 ft. will cut a brecciated chert carrying great quantities of water at considerable pressure.

RECENT GEOLOGIC DEVELOPMENTS ON THE MESABI IRON RANGE, MINNESOTA.

BY J. F. WOLFF, E. M., DULUTH, MINN.*

During the past four or five years, much has been added to the detailed geologic knowledge of the Mesabi Range. This has not been in the direction of discovery of any new fundamental facts, but of detailed study, sub-division and correlation of different parts of the whole formation and of individual orebodies. Prior to this time, mining engineers in the district were so engaged with the commercial interests of exploring and developing orebodies that close geologic study and sub-division was done in only a few instances. The demand for refinements of work in this direction has caused extensive structural sub-division and correlation to be done in all exploration work during the past four years by the engineers of the Oliver Iron Mining Company. Such work has become a commercial necessity rather than a scientific refinement, and at the present time is being extended to all parts of the range.

In the summer of 1914, the author of this paper wrote a series of articles on the orebodies and special features of the Mesabi Range, which was published in the Engineering and Mining Journal, issues of July 17 to August 7, 1915. Since that time studies of sub-division of the iron-formation have been extended considerably. The principal feature of this paper is the presentation of the sub-divisions of the iron-formation. To this is added a discussion of the relations of the orebodies to the gentle folding and fracturing of the formation, and special features of the range.

The outline will be as follows:

1. General geology.
2. Sub-division of Biwabik iron-formation.

*Mining Engineer, Oliver Iron Mining Co., Duluth, Minn.

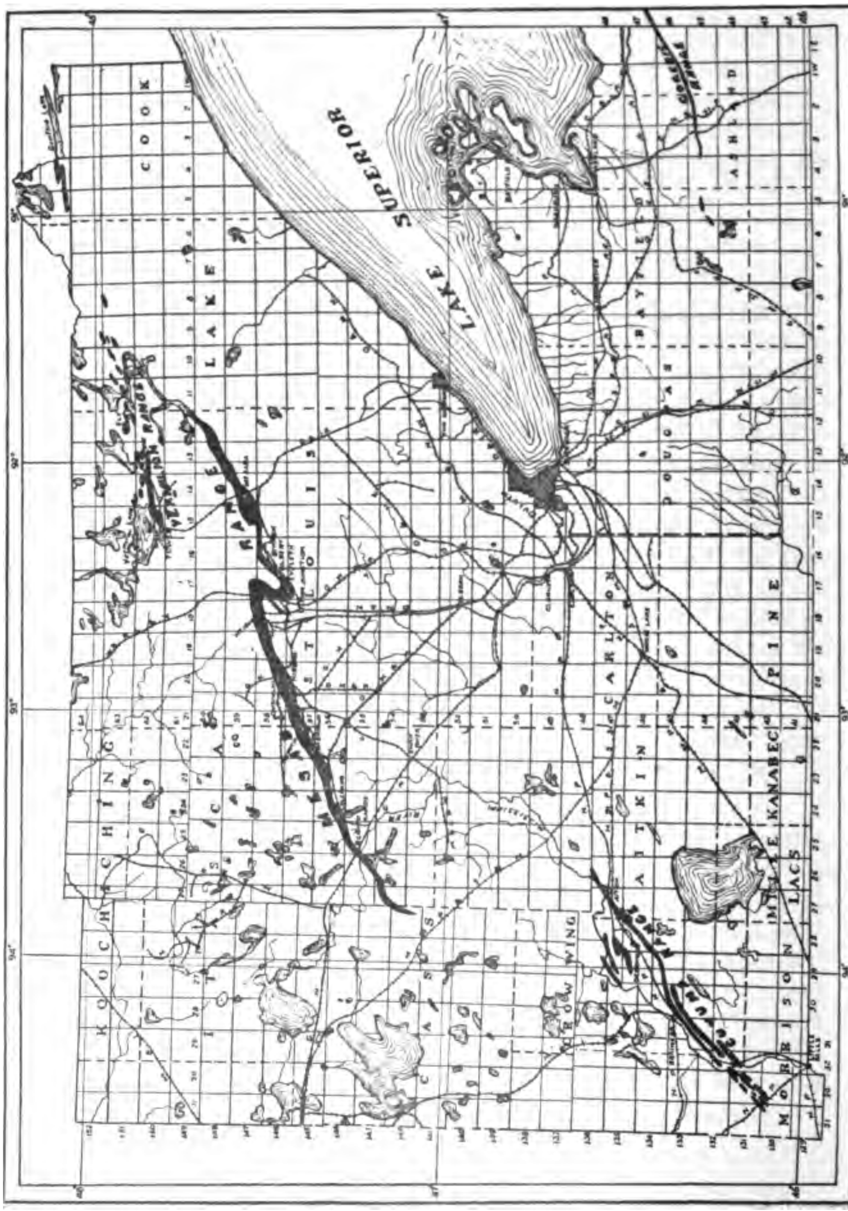

LAKE SUPERIOR
COOK
LAKE
CARLTON
AITKIN
PINE
KANABEC
MILLE LACS
DOUGLAS
CROW WING

Figure 1

3. Relation of orebodies to folding and fracturing of the iron-formation.

4. Special features.

Acknowledgment is due to W. J. Olcott, President, and John Uno Sebenius, General Mining Engineer, of the Oliver Iron Mining Company, for permission to use the information presented in this paper.

General Geology.

Correlation—Although the general geology of the Mesabi Range has been described most exhaustively in U. S. Geological Survey Monographs Nos. 43 and 52, and elsewhere, a short synopsis may be a necessary preliminary to the detailed discussion of the range presented in this paper, because many of the readers of this article may be unfamiliar with the above publications or the general geology of the Mesabi district.

The topographic feature of the Mesabi Range is a line of fairly prominent and continuous hills ranging in elevation from 1,400 to 1,900 ft. above sea level, composed of a complex of granites, green-stones, green schists, slates, graywacke and conglomerate. Resting uncomformably upon this basement complex, and sloping away at a gentle angle to the southeast, is a series of sedimentary rocks, the middle member of which is iron-bearing. The outcrop of this iron-bearing member is the geologic feature known as the Mesabi (or Missabe) Iron Range. Within this formation the iron orebodies are found. Its outcrop has been traced by explorations from Sec. 12, T 142 N., R. 25 W., northeastward to Birch Lake, in Sec. 26, T. 61 N., R. 12 W., a distance along the strike of about 112 miles. Its width varies from 3/8 to 3 miles, due to variation in the dip and thickness. The sedimentary series above mentioned consists of a basal quartzite, named Pokegama, an intermediate iron-formation, named Biwabik, and a top black slate, named Virginia slate.

The location and general extent of the range is shown on the map, Fig. 1. The general relations of the iron-bearing member to associated rocks are shown by the cross-section, Fig. 2, better than further description could explain.

Instead of the complete correlation table as given by the U. S. Geological Survey, a simplified geologic column is herewith given, which is adequate for the engineer in the district.

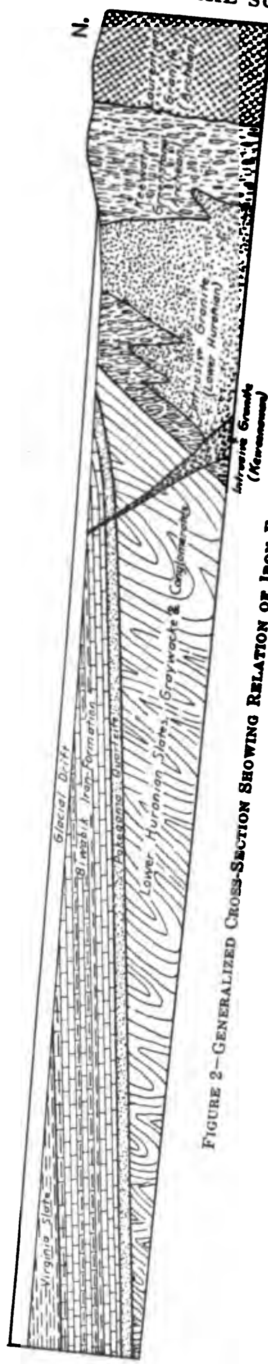


FIGURE 2—GENERALIZED CROSS-SECTION SHOWING RELATION OF IRON FORMATION TO ASSOCIATED ROCKS

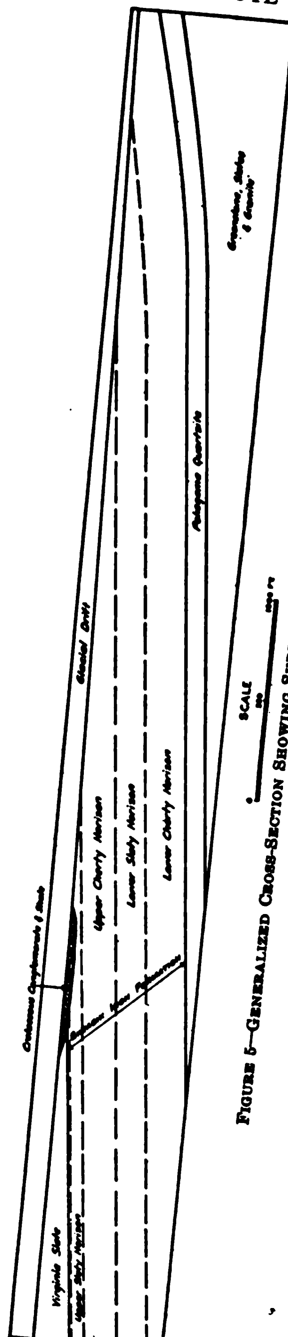


FIGURE 5—GENERALIZED CROSS-SECTION SHOWING SUBDIVISION OF THE BIWABIK IRON-FORMATION

Quaternary System Pleistocene Series	Glacial drift, 0 to 300 ft.	
Unconformity.		
Cretaceous System	Conglomerates and shales, 0 to 50 ft.	
Unconformity		
Algonkian System .... Keweenawan Series	Duluth Gabbro Embarrass Granite	East end of range.
Unconformity Upper Huronian Series. (Sedimentary)	Virginia slate 0 ft. to great thickness. Biwabik Iron-Formation, 475 to 775 ft. Pokegama Quartzite, 50 to 150 ft.	
Unconformity Algonkian and Archean Series...	Basement complex of Slate-graywacke-conglomerate series, granites, greenstones, green-schists and porphyries.	

From a commercial standpoint the Upper Huronian series, Virginia Slate, Biwabik Iron-Formation, Pokegama Quartzite are the important rocks of the district.

The characteristics of the different formations need not be discussed. This will be done for the iron-formation later on.

OREBODIES.

The Upper Huronian Quartzite-Iron Formation Slate series is an ordinary (except for the iron) series of elastic sediments, deposited in fairly shallow water, contemporaneous with the middle member of which was deposited or precipitated out of solution an enormous quantity of iron. At the present time by far the greater part of iron, even in the fresh, deeply buried, original rock, is in the form of iron oxide, either hematite, magnetite, or an oxide intermediate between them. The iron occurs as thin lenses and shoots from a fraction of an inch to a few inches thick, as irregular masses, and as granules varying in size from one-tenth of an inch diameter to a pin point, cemented together in an amorphous silica or chert matrix. A minor amount of green granules of iron-silicate, called greenalite, is found inter-bedded with slate layers described below and intimately intermingled with the lenses, masses and granules of iron-oxide. Geologists of the U. S. G. S. have concluded from field and laboratory evidence that practically all of the iron was ferrous-silicate, greenalite, originally, and that the oxidation took place either very rapidly after the precipitation or deposition of the greenalite or at a later period.

The writer does not agree with this entirely. Thousands of feet of drill cores examined give no evidence that the lenses of iron-oxide which make up the great bulk of the iron in the formation ever have been in any other than their present chemical state since they were laid down. Many of the granules of iron may have been greenalite and have been oxidized to hematite or magnetite, but it is hard to account for the most intimate intermixture of hematite and fresh greenalite grains in a solid fresh rock core which shows no evidence of leaching or secondary oxidation. The writer believes that the great mass of iron was laid down as original oxide (hematite and magnetite or an intermediate oxide), cemented together in an amorphous silica matrix. While it is true that under the microscope numerous oxidized greenalite granules can be seen, it does not follow that all the oxide grains were greenalite originally. The intimate association of hematite grains and fresh greenalite granules is not explained, in fact seems to be counter to this theory of oxidation from original greenalite. Moreover, the greatest mass of iron in the original rock being in the form of thin layers and lenses of iron-oxide does not require any step process from an iron silicate to the oxide to explain its occurrence. In fact, the silicate theory is counter to the evidence in this particular, so far as the writer can see. In short, the writer is convinced that by far the greater quantity of the iron in the original Biwabik formation was deposited as an original oxide,—hematite and magnetite, or an intermediate oxide. Most of the cores can be picked up or rolled along a table with an ordinary small horseshoe magnet. Determinations made on some of the magnetic oxide lenses have shown that, after crushing to pass 100 mesh, 71 per cent. of the material could be separated out magnetically. The original sample analyzed 52.28 per cent. dry iron, the magnetic portion 65.30 per cent. dry iron, and the non-magnetic portion 21.94 per cent. dry iron. Of the magnetic portion 24.32 per cent. was ferrous oxide. In pure magnetite 31 per cent. of the mineral is ferrous oxide Of the non-magnetic portion 15.70 per cent. was ferrous oxide. Of course the crushing may have induced some magnetic susceptibility.

The lenses and layers of iron oxide in the fresh rocks are very uniformly fine grained in texture, although occasionally crystallization can be seen. In many cases original shallow water structures can be seen on their bedding planes or at contacts with chert lenses. Much of the fine iron in the well banded parts of the formation and in some of the cherty taconite is 100 mesh fine or finer and presents the appearance, especially in the latter case, of having been scattered in the rock as if by a pepper shaker.

No denial of the existence in the iron formation of large quantities of greenalite is made here,—only its relative importance is disputed. When the facts above cited and the evidence available to the writer are considered by the Lake Superior geologists of the U. S. G. S., no doubt they will agree in placing greater emphasis upon the original oxides and less upon the silicate, as the original form of the iron in the Biwabik formation. The enormous quantities of original iron-oxide in Brazil have quite altered our previous ideas as to the origin of high-grade iron-oxide orebodies.

Iron carbonate, grunerite, amphibole and actinolite occur in small quantities in the Biwabik formation.

The bedding planes of all three members of the Upper Huronian series are approximately parallel. Interbedded with the iron formation in certain horizons are numerous slate layers, varying in thickness from a few inches to many feet, and a considerable quantity of conglomerate, some of the pebbles of which are iron ore, some are red chert or jasper, and some are derived from the older granites and greenstones. Some contorted jaspery phases occur also, as does a considerable quantity of granular red material, resembling oolitic grains or a red jasper sand. In the ore bodies this latter material can be found, decomposed, but otherwise unaltered. The iron-formation from Virginia slate to quartzite varies in thickness from 475 ft. to 775 ft., the average being about 600 feet.

After this series of sediments had been laid down and solidified, earth movements raised them above water level, allowing erosive agencies to cut through the overlying slate into the underlying formations. These earth movements warped the formations and cracked the brittle iron-formation quite extensively, allowing surface waters to enter its upturned edges. Especially where such cracking was pronounced, the ground-waters entering the iron-formation, carrying carbon dioxide in solution, attacked the ferrous iron compounds and oxidized them. In such localities much of the ferrous silicate has been changed to hematite, the hematite now occurring as disseminated particles in the chert, the rock still retaining its solidity. The whole iron-formation, whether thus altered or not, is a ferruginous chert with interbedded slate layers and locally is called "taconite." This term should apply strictly to the ferruginous chert and not to the slate layers, though in some horizons the slate bands and chert layers are so intimately interbedded as to make a distinction quite impossible.

Where the cracking has been most intense the circulation of ground-water has been most vigorous, the solvents in the ground-waters have leached out the silica and other minor constituents of the rocks and have left in place the original and secondary iron-oxide. Such residual material now constitutes the orebodies. They are surrounded on all sides by the rock walls of the iron-formation from which they are derived. Pore-space was developed by this removal of silica and the settling or slumping in place of the layers of the orebodies is a characteristic feature. The typical orebody thus developed has a

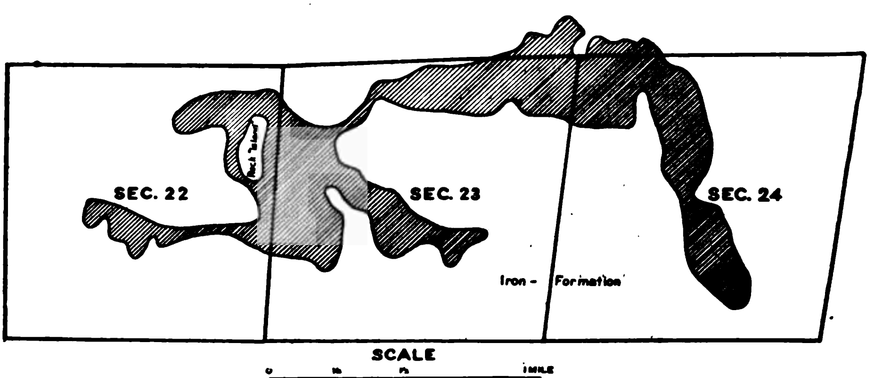


FIGURE 3—PLAT SHOWING SHAPE AND AREA OF TYPICAL OREBODIES

trough structure and an irregular trough shape. Orebodies vary in size from a few acres to several hundred acres, and from a few feet to several hundred feet in thickness. In many places small troughs unite to form a large one. Fig. 3 shows the shape and area of a typical orebody. While the trough orebody is the typical one, there are two other types, namely, the flat-layered body and the fissure-type orebody. The former is either the remnant left by the erosion of a former trough-body or it is an ore layer continuing down the dip from a trough-body. Usually such a layer has a rock (slate) capping. The fissure-type orebody is an incompletely developed trough-body and is usually associated with a larger trough orebody. Fig. 4 shows the development of a trough

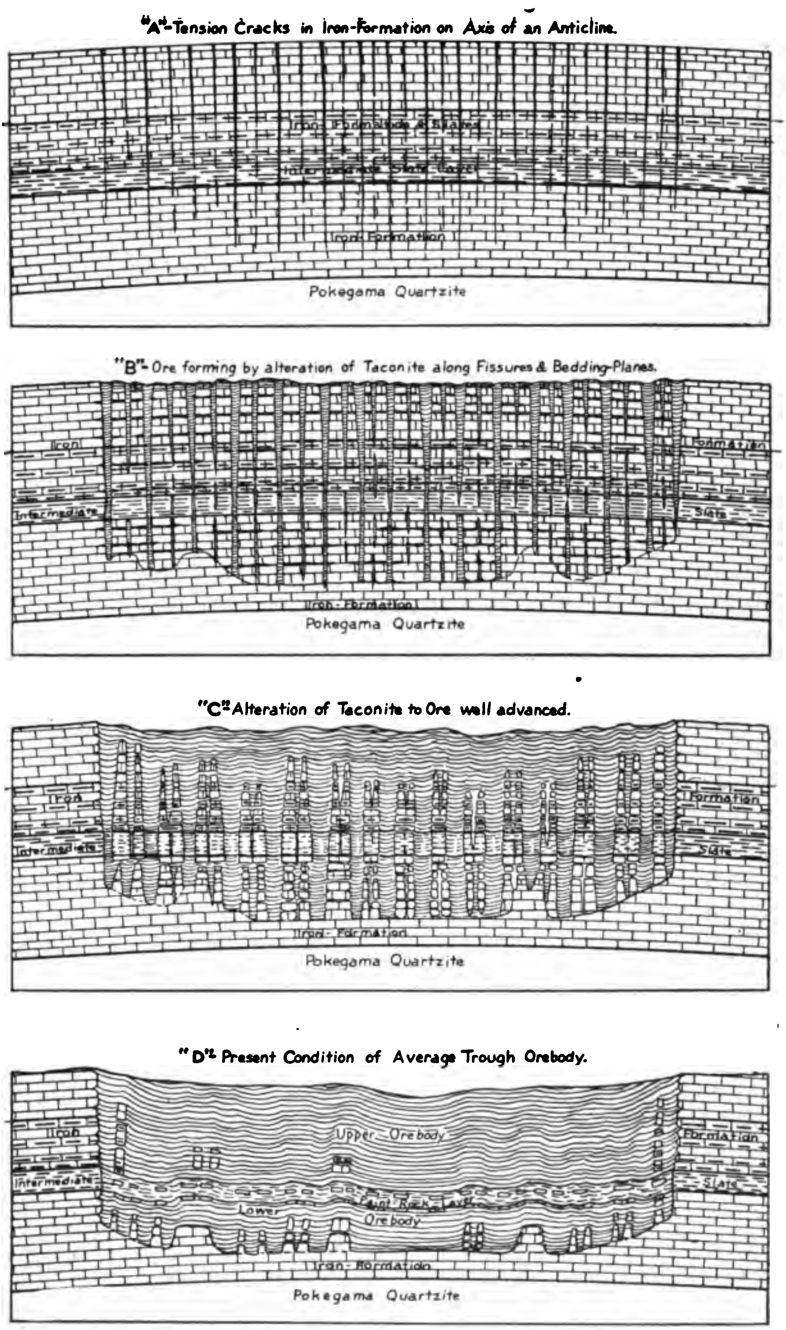


Figure 4—Cross-Section Showing Stages in Development of a Trough Orebody on the Axis of a Gentle Anticline

orebody on the axis of an anticline, where cracking has been pronounced. All four stages here shown can be observed in the field. Notice the slumping of ore layers near the rock walls. The features of the three different types of orebodies were discussed at length in the Engineering and Mining Journal, July 17 to August 7, 1915, and will not be repeated here.

Covering the entire iron-formation, except at a few isolated spots, is a mantle of glacial drift, varying in thickness from a few feet to 300 feet.

Sub-Division of the Iron-Formation.

Detailed Sub-Division—The item of principal interest and most practical value to mining men in the Mesabi district, presented in this paper, is the sub-division of the iron-formation into its several horizons. As indicated above, the engineers of the Oliver Iron Mining Company have found it absolutely essential to do this kind of work in the exploration and development of their orebodies. Undoubtedly others will meet with the same experience. The determination of the structure of an orebody depends upon the proper sub-division and correlation of the different ore and rock layers in the drill records.

Such correlation, not only in one area but over the length of the range, has shown that the formation can be sub-divided into four principal horizons, each of which has remarkably uniform characteristics from one end of the range to the other. Each horizon can be sub-divided further. Of course, the thickness of each horizon is not uniform because the thickness of the whole formation varies. There are local peculiarities in some places, but these do not destroy the major sub-divisions. This sub-division and correlation has been done chiefly by F. B. Cronk, Mining Engineer for the Oliver Iron Mining Company, and the writer.

Fig. 5 shows the four sub-divisions which can be made in any part of the Mesabi district. From the top down they are, Upper Slaty Horizon, Upper Cherty Horizon, Lower Slaty Horizon and Lower Cherty Horizon. They are named from the predominant physical characteristic of the rock in them. Commercially the two cherty horizons and Lower Slaty Horizon are most important, as the principal orebodies occur in them. The two cherty horizons contain very few slate seams, but the slaty horizons contain a considerable number of interbedded chert layers, and a great amount of greenalite. The lower half of the Lower Slaty Horizon contains the most slate, the bottom 30 ft. or so being almost pure slate. But the upper half is slaty or banded in structure rather than in composition of the rock layers. The sub-divisions are made by eye rather than on any chemical basis or microscopic examination.

The Upper Slaty Horizon is composed about half and half of slate layers and greenalite and iron-oxide lenses. The upper part approaching the Virginia slate is the more slaty. It is separated from the Virginia slate by a layer 10 or more feet thick of a carbonate rock, probably an impure limestone.

Two cross-sections, Figs. 6 and 7, are presented to show in detail the four main sub-divisions, the minor sub-divisions of each, the relations of the orebodies to these horizons and the kind of ore derived from each.

Fig. 6 shows an east-west cross-section, looking north, through the Adams, Hull, Nelson, Leonidas orebodies just west of Eveleth. It is the best cross-section that has been or can be made from present explorations, so far as the author knows. The location of the section is shown on the accompanying map, Fig. 11, and is approximately at right angles to the strike. It is developed from drill-hole classifications, as can be seen, but practically the entire section of ore is being developed now by open-pit and underground workings. Because this cross-section is taken along the longitudinal axis of the ore trough, the trough structure does not show. In the area of drill holes 6 to 8 inclusive, a north-south trough is tributary to the east-west trough through which this cross-section is taken, and the trough structure is apparent. The gentle warping of the entire series can be noted. The principal warping is at right angles to the strike, however.

The relations between the different horizons and their derivative ores are so evident from the cross-section that a detailed description need not be given. In the area of drill holes 2 to 15 inclusive, the taconite in the Lower Slaty and Upper Cherty Horizons is so badly altered that classification is very difficult.

A few features may call for comment. The black slate indicated at the base of the Lower Slaty Horizon is the so-called "Intermediate" Slate, which makes the characteristic paint rock layer of the typical Mesabi orebody. The orebodies

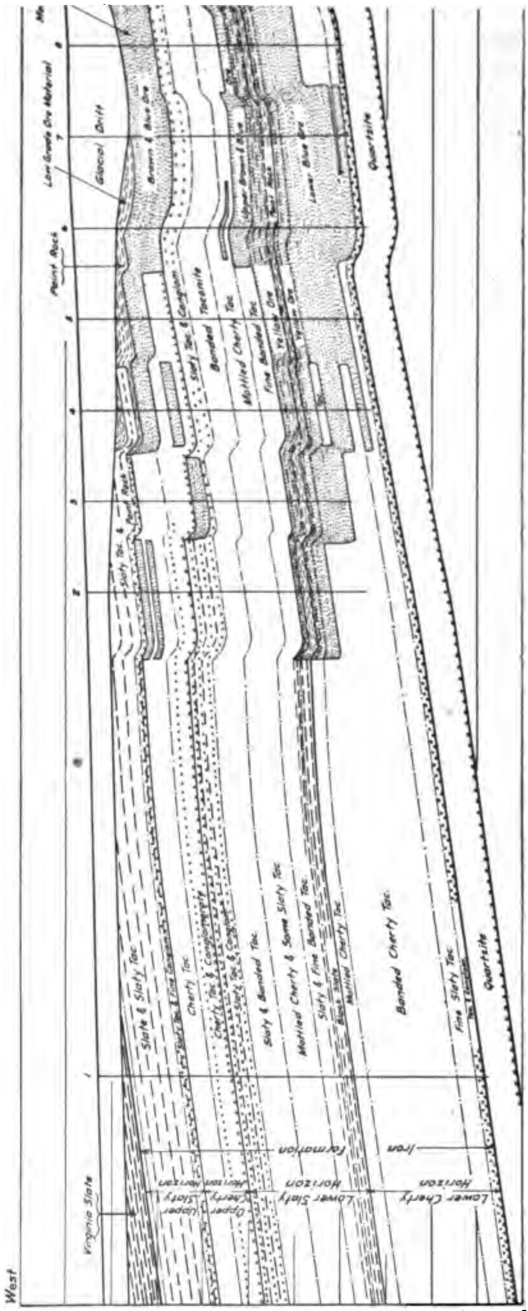


FIGURE 6—LONGITUDINAL CROSS-SECTION OF A LARGE TROUGH OREBODY SHOWING ORE DERIVED FROM ALL HORIZONS OF THE IRON FORMATION
(Continued on two succeeding pages)

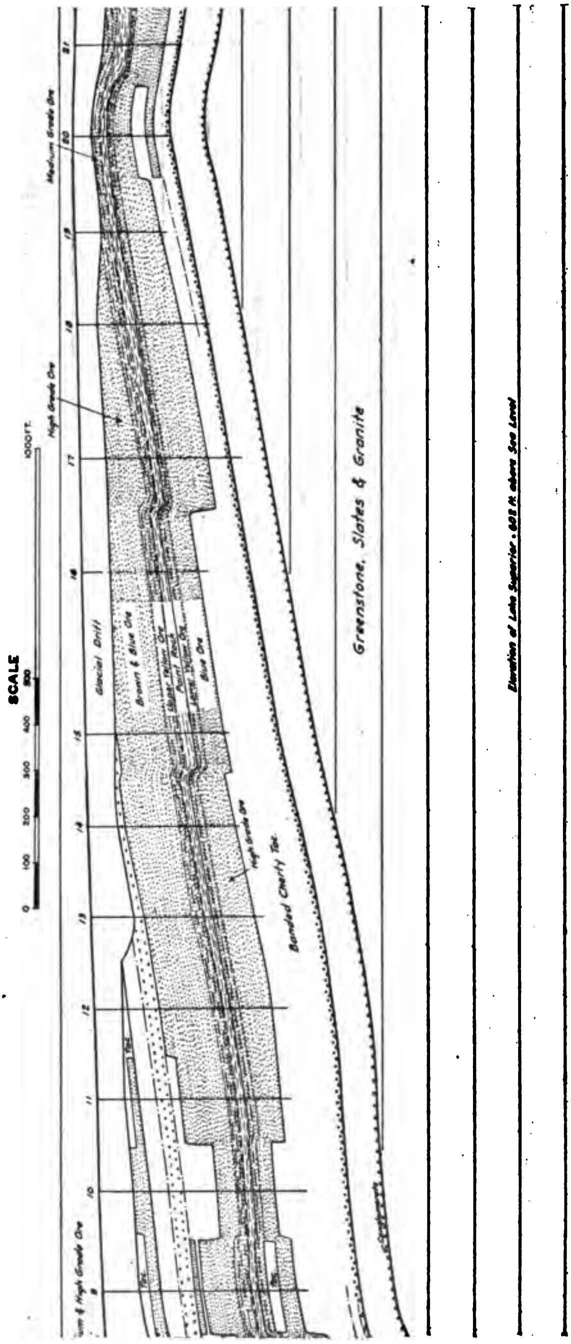


FIGURE 6—(Continued)

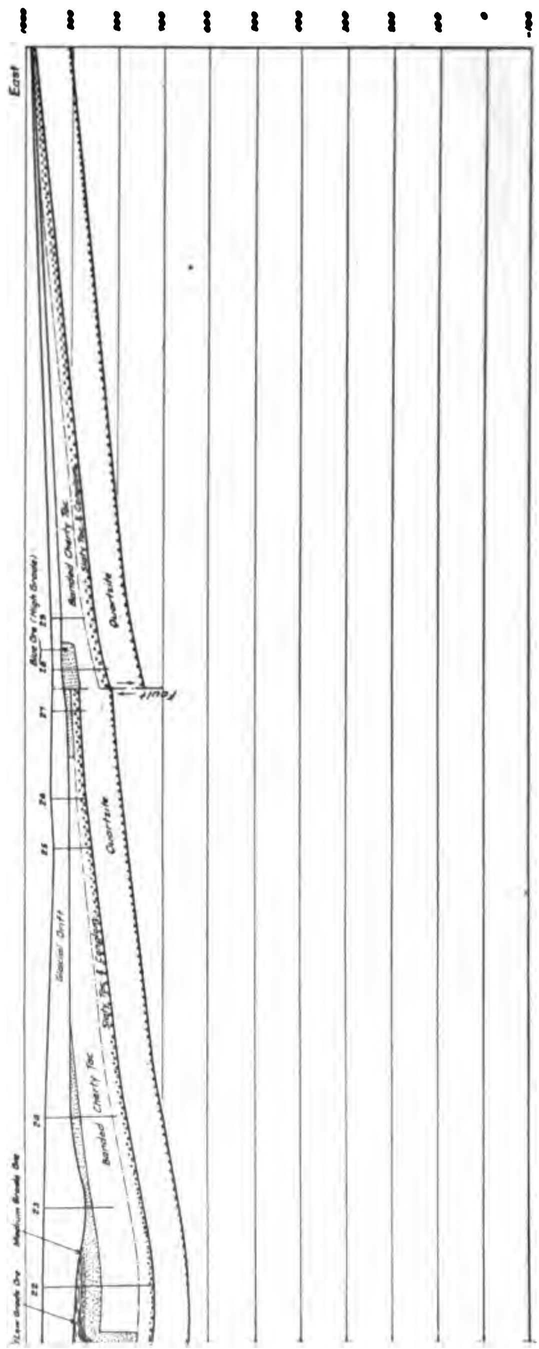


FIGURE 6—(Continued) LONGITUDINAL CROSS-SECTION OF A LARGE TROUGH OREBODY SHOWING ORE DERIVED FROM ALL HORIZONS OF THE IRON FORMATION

first developed on the range were located in that part of the formation shown between drill holes 15 and 24; therefore, it is easily seen that the typical orebody had five layers—upper and lower blue ore layers, upper and lower yellow ore layers, and an intermediate paint rock layer.

Interbedded in the Upper Cherty Horizon and the top of the Lower Slaty Horizon are distinct conglomerates. At the base of the Upper Slaty Horizon there is also a fine interbedded conglomerate. The Upper Cherty conglomerate is continuous from the far eastern part of the range to the western part. It was recognized in drill cores only a few years ago by W. H. Crago, head of the exploration division of the Mining Engineering Department of the Oliver Iron Mining Company. All of the earlier drill cores have not been re-examined for it as yet, but it has been correlated extensively in different parts of the range by F. B. Cronk and the writer. The cross-section in Fig. 6 shows a maximum development of interbedded conglomerate It is not as thick either on the western or eastern end of the range. On the eastern part it is distinctly developed in both Upper Cherty and upper part of Lower Slaty Horizons, but in the west central part of the range only the Upper Cherty conglomerate has been recognized, and it is quite thin. This section (Fig. 6) shows practically a maximum thickness of iron-formation. The Lower Slaty Horizon is abnormally thick (260 ft) whereas the average thickness is only about 150 ft. In the central and western part of the range, the Upper Cherty Horizon is exceptionally thick, the slaty layers of the upper part of the Lower Slaty Horizon being replaced by cherty and banded taconite. Evidently, more muds were deposited with the iron-formation in the east central part of the district than in the central and western parts of the district. Perhaps the underlying rocks outcropping to the north are accountable for this. From Mt. Iron east to Aurora, large areas of greenstones and slates lie to the north of the iron-formation. If the original shore line of the sea in which the iron-formation was laid down occupied approximately its present position, the weathering and erosion of these rocks contributed muds and argillaceous sediments to the sea water contemporaneous with the deposition of iron.

Fig. 7 shows a cross-section across a typical trough orebody in the Virginia district. The Upper Cherty and Slaty

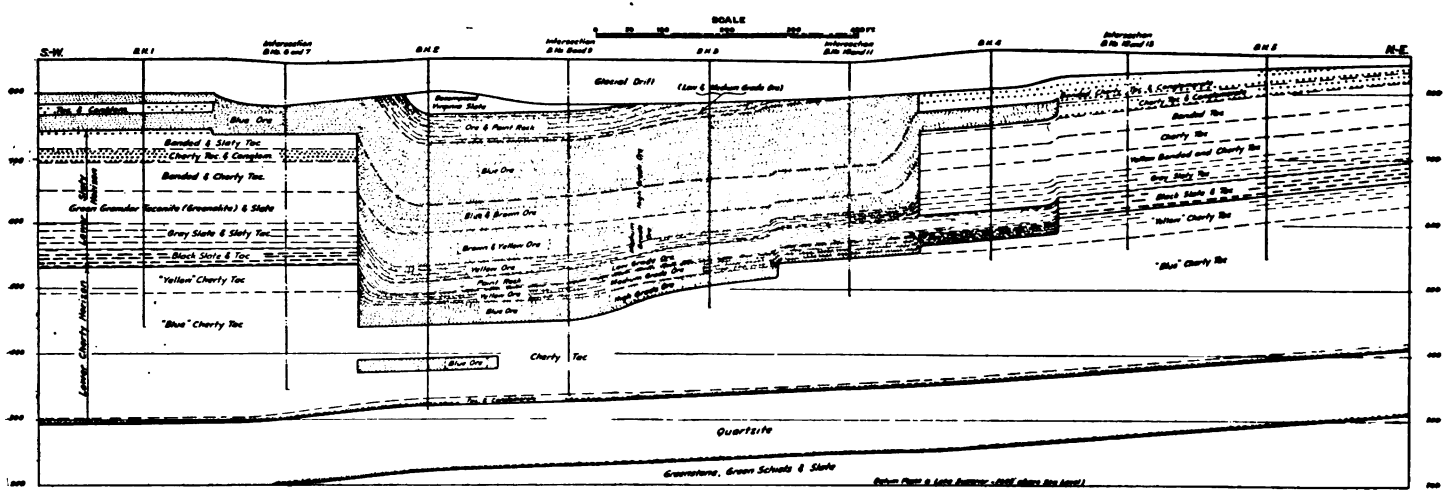


FIGURE 7—TRANSVERSE CROSS-SECTION OF A TROUGH OREBODY SHOWING ORE DERIVED FROM THE DIFFERENT HORIZONS AND TROUGH STRUCTURE OF OREBODY

Horizons have been eroded from the sides of the rockwalls of the orebody, but all horizons from the Virginia slate down are represented in the orebody. The typical trough structure is well shown.

Records and Sub-Divisions of Drill Holes in Different Districts—Fig. 8 shows records and sub-divisions of drill holes in the Nashwauk, Hibbing, McKinley and Aurora districts, comprising the territory from the west central to the eastern part of the range. These records all show the same major divisions, though varying in the minor sub-divisions and dimensions. The interbedded conglomerate in the Upper Cherty Horizon is persistent in all the districts.

Ores Derived from the Different Horizons—The characteristic ore derived from the Lower Cherty Horizon is a coarse "blue" high-grade ore. It contains practically no paint rock seams. About 30 or 40 ft. of the top of this horizon is a brown or yellow mottled cherty taconite (originally a greenalite rock) containing some slate seams in its upper measures, and this layer makes a yellow ocherous ore of medium grade. At the base of the Lower Cherty Horizon just above the basal conglomerate is a layer of fine slaty taconite (see Fig. 5) which makes a yellow ocherous ore. With the exception of these top and bottom layers, the Lower Cherty Horizon makes a "blue" high-grade ore. As used here, high-grade ore means ore averaging about 59 per cent. dry iron, medium-grade averaging 55 to 56 per cent. dry iron, and low-grade averaging about 50 per cent. dry iron.

The characteristic physical feature of ores from the Lower Slaty Horizon is their finely banded and slaty texture. As previously indicated, the black slate at the base makes the paint rock layer so prominent in every typical orebody. This material is not a commercial ore. It is highly aluminous and contains 20 per cent. or more of moisture. The gray slate and greenalite and slate above this black slate (Fig. 7) make a medium-grade yellow and brown ore, the yellow ore being quite aluminous. The banded-cherty and banded-slaty taconite of the top of the Lower Slaty Horizon make a very fine-grained blue and brown ore of high grade. It was this kind of ore which was so objectionable to furnace men because of excessive fines, in the early days of the Mesabi Range exploitation.

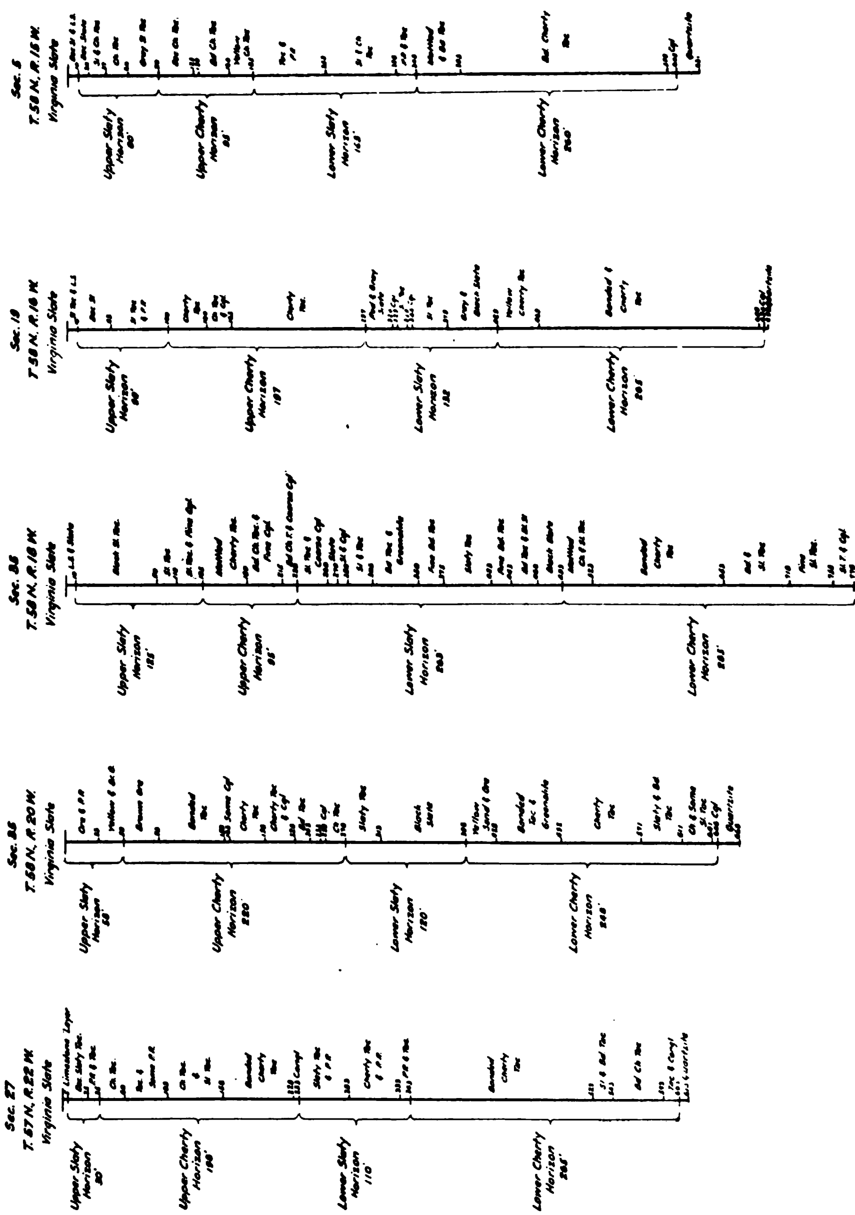


FIGURE 8—CHART SHOWING RECORDS AND SUBDIVISIONS OF DRILL HOLES IN DIFFERENT DISTRICTS ALONG THE MESABI RANGE

The Upper Cherty Horizon makes a high-grade coarse blue ore, in all of the large well-concentrated bodies. It is indistinguishable in texture from the blue ore of the Lower Cherty Horizon.

In some orebodies toward the south side of the formation, such as the Morton Mine in the Hibbing district, the Duncan in the Chisholm district, the Leonidas in the Eveleth district and the Schley and Hobart in the Gilbert district, the ore in this horizon is somewhat sandy and cherty, due in part to incomplete concentration and in part to secondary silica deposited by ground waters.

The Upper Slaty Horizon generally makes a low-grade non-merchantable ore. The one known exception to this is shown in Fig. 7, in which orebody most of this ore probably will be merchantable. This orebody, however, is one of the most highly concentrated on the range. In most orebodies which have this Upper Slaty member, the material from it is a soft plastic paint rock with decomposed chert and greenalite layers. It resembles very much the so-called "Intermediate" paint rock layer.

From the cross-section, Fig. 6, it is evident that all orebodies will not contain all of these horizons or layers. The unit of land sub-division is a 40-acre tract and many mines occupy one or a part of one such tract. If a mine is located near the quartzite outcrop, most of the upper horizons will have been eroded away, and as the mine location approaches the Virginia slate outcrop more of the upper layers will be found in the orebody.

Relation of Orebodies to Folding and Fracturing of the Iron-Formation.

In the Engineering and Mining Journal articles above referred to, the author stated that the data then at hand indicated that the orebodies have formed in places where gentle folding and warping of the iron-formation had fractured it considerably, allowing easy access and circulation of ground-waters. Evidence along this line has been assembled since the publication of those articles and in every case where the exploration data is complete enough it has been found that the orebodies occur where the whole formation has been warped. In the eastern part of the district (Virginia and eastward) the

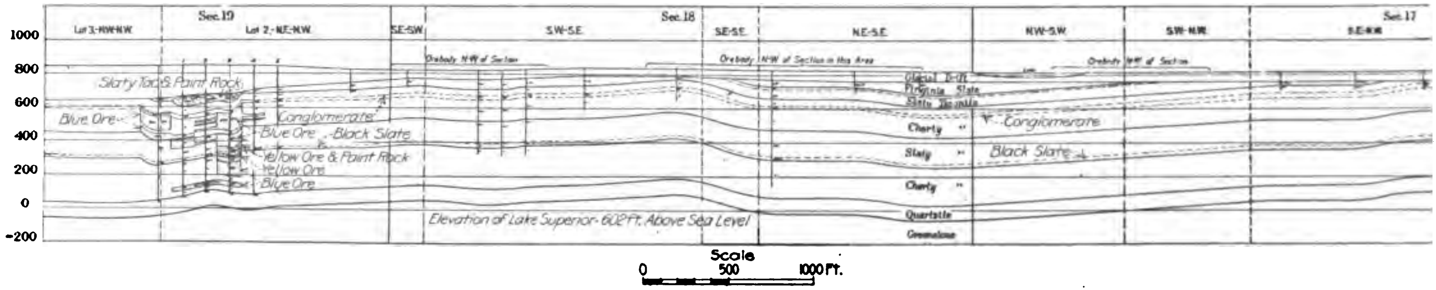


Figure 9—Cross-Section Parallel to Strike of Range, Showing Subdivisions of Iron Formation and Relation of Orebodies to General Structure

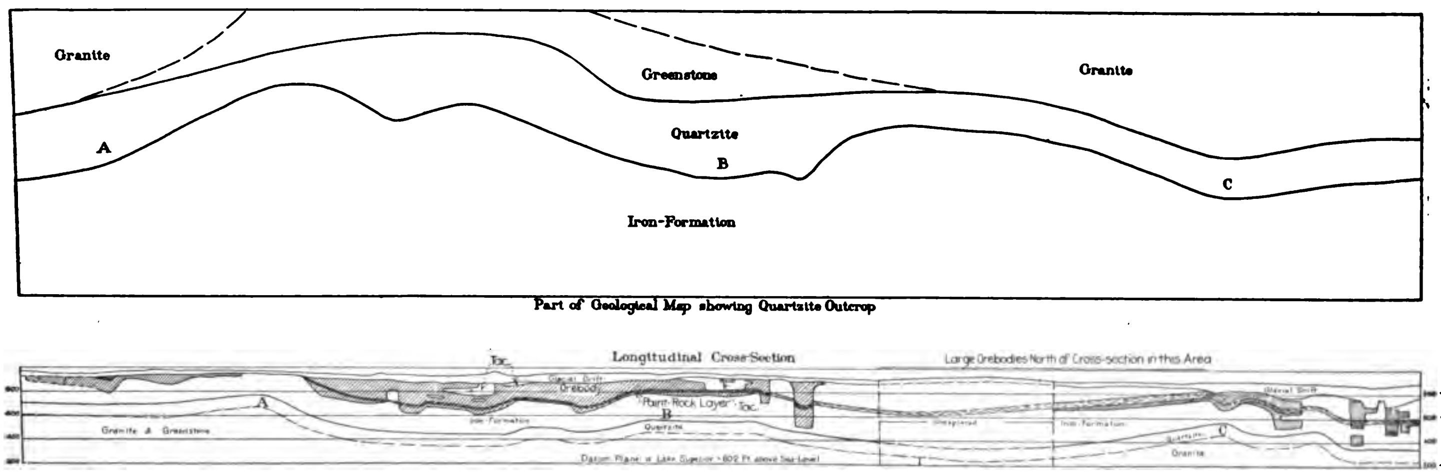


Figure 10—Part of Geologic Map and Cross-Section Showing Relation of Orebodies to General Structure of Entire Iron Formation in Hibbing-Chisholm District

orebodies are on the crests of gentle anticlines or on axes of combined anticline and synclines. Fig. 9 shows a cross-section parallel to the strike of the formation in T. 58 N., R. 16 W., location of which is shown on the map in Fig. 11. This section is taken well to the south side of the formation and only one orebody reaches it. But the locations of other orebodies northwest of it are shown on the cross-section, and in each case where the exploration data is complete enough to show it, the orebody is located on the axis of an anticlinal flexure or combined anticline, and syncline.

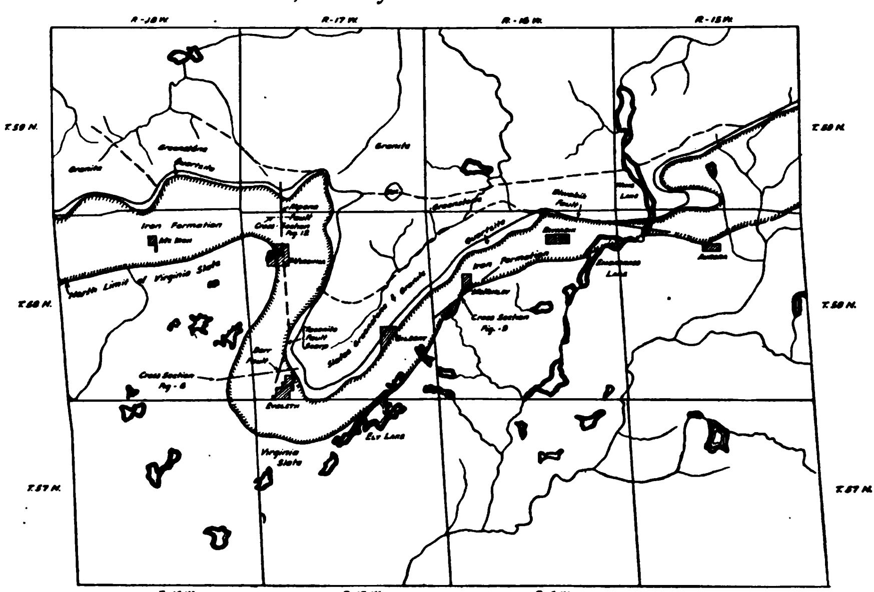


FIGURE 11—PLAT SHOWING AREA OF EAST CENTRAL MESABI RANGE

In the central part of the range, great broad flexures rather than merely localized ones seem to have determined the locations of the orebodies. The formation was very generally cracked up and the broad structural basins directed the flow of underground waters. Fig. 10 shows a structural cross-section taken midway between Virginia slate and quartzite outcrops approximately parallel to the strike, through the Hibbing-Chisholm districts. A part of the quartzite outcrop to the northwest is also shown. Three prominent anticlines, A, B and C, with two intervening synclines, are shown. The cross-section is taken about a mile south of the quartzite outcrop. It has been published previously in the Engineering and Mining Journal, August 7, 1915. It shows that the orebodies occur quite continuously over both crests of anticlines and troughs of synclines, indicating a very general fracturing of the formation, vigorous circulation of ground-water and consequent complete alteration and concentration of iron-formation. These major flexures can be determined only by such correlation and drawings as are shown in the two cross-sections, Figs. 9 and 10, but the minor flexures within the larger ones often can be observed in the field.

Special Features.

Special features of the Mesabi Range may be of interest and deserving of inclusion in this paper.

Faults.

The two principal faults known to date on the Mesabi Range are those known as the Biwabik and Alpena faults, both of which were described in the Engineering and Mining Journal, July 24, 1915. Mention is made of them here only because they have been followed further since that time. Fig. 11 shows the location of both. The Biwabik fault has been traced to the NW of SE, Sec. 5, 58-15, where it practically disappears. It is a hinge-type gravity fault, the south side of which has been depressed. The greatest throw is at its west end at the Biwabik Mine, Lot 4, Sec. 2 and Lot 1, Sec. 3, T. 58 N., R. 16 W., where the vertical displacement exceeds 200 feet. The underlying greenstone is faulted up against the ore, though the faulting probably occurred prior to the formation of the ore. Fig. 12 is a cross-section of the Alpena orebody north of Virginia, showing the largest fault known on the range. The location of the cross-section is shown on Fig. 11. As indicated, the strike of the fault is approximately north and south. It is a faulted thrust-fold, the probable development of which is shown by Fig. 13. The genesis of the fault is directly connected with the gentle uplift or crustal warping which produced the large Z-shaped bend in the range known locally as the Virginia "horn." This was discussed also in the Engineering and Mining Journal, July 24, 1915. It is repeated here because since that series, a fault (shown on the east end of cross-section in Fig. 6) has

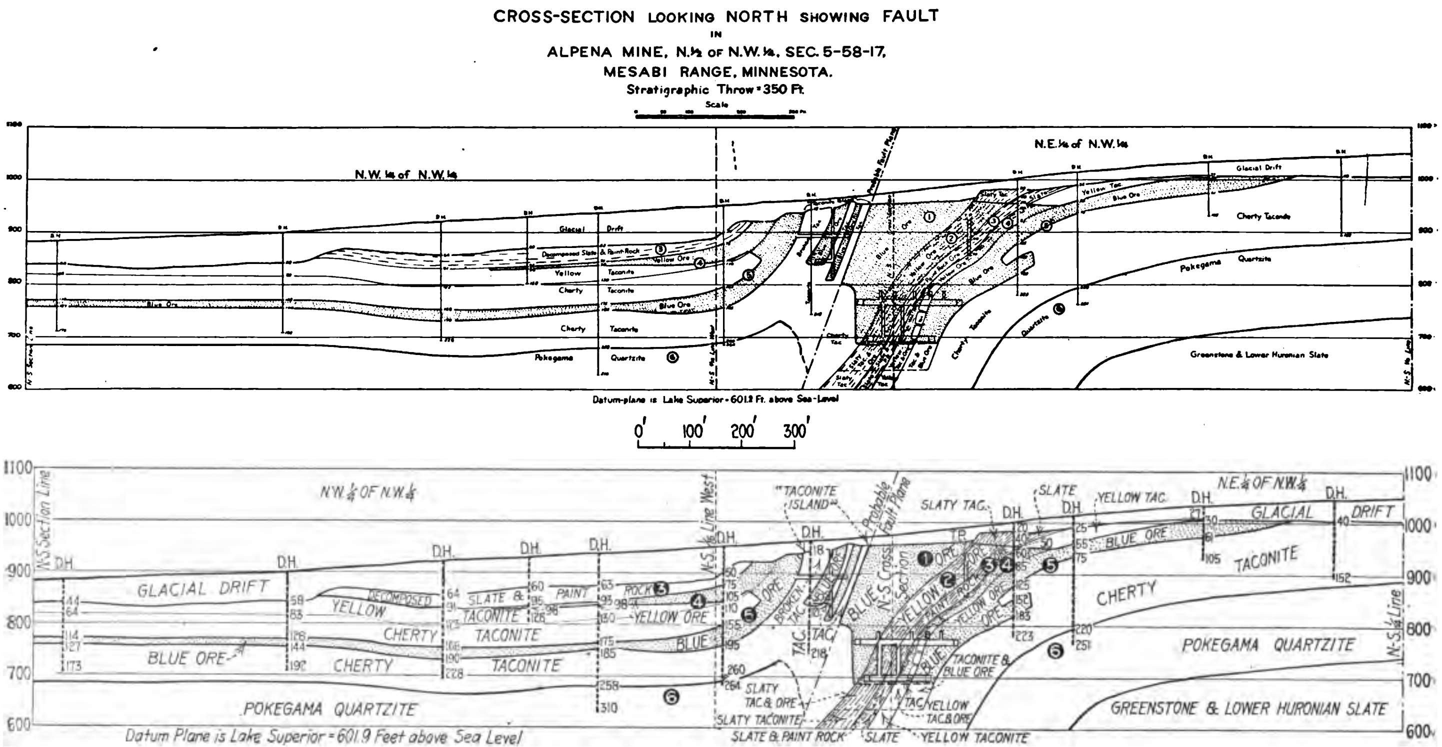


FIGURE 12—SHOWING ALPENA THRUST-FAULT

been discovered by correlation of drill holes. The location of this fault, which may be called the Dorr fault from the property on which it is located, is shown on the map, Fig. 11, and is of the same type (reverse or thrust) as the Alpena fault. It is almost directly south of the Alpena. Between the two and about ¾ mile north of the Dorr fault is a taconite bluff, the east side of which is a very steep wall, un-

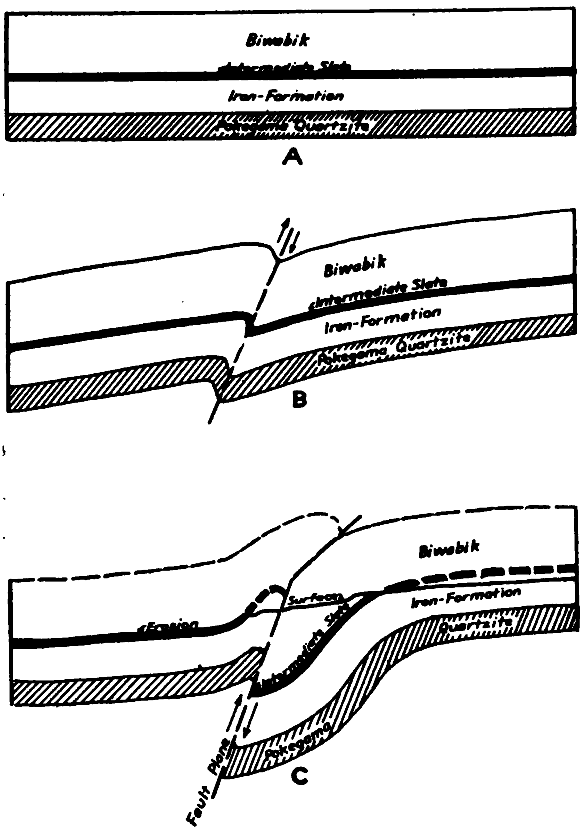


Figure 18—Showing Probable Development of the Alpena Thrust-Fault

doubtedly a fault scarp. It is of the same type as the Dorr and Alpena faults. Although we have as yet no complete exploration data, it appears that the Alpena and Dorr faults and the intervening escarpment are one continuous fault produced by the crustal movements which caused the Virginia "horn." This probable connection is indicated on Fig. 11. However, it is possible that the Alpena fault is entirely separate from the Dorr fault.

The average thickness and dip of the whole iron-formation in different parts of the range may be of interest. So far as they have been determined from the present very extensive explorations, they are shown in the accompanying table, tabulated by ranges.

It will be noted that west of the Alpena fault the dips are quite low and uniform, while east of it they are higher and rather irregular. From this evidence and that of the Alpena-Dorr fault, it appears that the disturbance caused probably by the intrusion of the great mass of Duluth gabbro between Lake Superior and the eastern Mesabi range tilted the eastern part of the range considerably; that the Alpena fault took up most of the crustal shortening in the Upper Huronian series due to the compression from the east; and that, because of the faulting, the sedimentary series west of it was relatively undisturbed.

	Average dip Degrees	Thickness Feet
T. 56 N.—R. 24 W.	6	520
T. 56 N.—R. 23 W.	3 to 8	475 (approx.)
T. 57 N.—R. 22 W.	5	615
T. 58 N.—R. 21 W. T. 57 N.—R. 21 W.	5	590
T. 58 N.—R. 20 W. T. 57 N.—R. 20 W.	4½	650
T. 58 N.—R. 19 W.	3 to 7	660
T. 58 N.—R. 18 W.	3	630 west of Alpena fault
T. 58 N.—R. 18 W. T. 58 N.—R. 17 W.	6 to 9	755 east of Alpena fault
T. 57 N.—R. 17 W.	10 to 15	
T. 58 N.—R. 16 W.	12	650
T. 58 N.—R. 15 W.	25	530 (Secs. 5 and 6)

The thickness is seen to be greatest in the vinicity of Eveleth and thinnest on the extreme eastern and western ends. The average of the figures given is about 620 feet.

Volumetric Shrinkage in Solid Taconite—Recent structural study has shown that the alteration which the original iron-formation has suffered has produced a volumetric shrinkage even where the iron-formation still retains its solid condition. However, in such solid taconite, hard layers of high-grade iron ore are interbanded with chert layers, the whole mass being firmly cemented together. Drill holes in fresh unaltered iron-formation alongside of holes in altered but still solid taconite show this fact to exist. Such a case is shown in comparing the depths of formation below the Lower Slaty Horizon in holes 1 and 2 with those in holes 4 and 5 in the orebody in Sec. 19, Fig. 9; also holes 1 and 2, Fig. 6.

This fact is a very important one to the mining engineer in the district in working out the structure of orebodies. The knowledge of it will aid him to establish more accurately the correct structure of certain ore layers. It explains the apparent lack of parallelism between different members of the iron-formation, which cross-sections often indicate unless this fact is known and applied.

Post-Algonkian Conglomerates on the Upper Huronian Series—In the Engineering and Mining Journal, July 17, 1915, the writer called attention to at least one conglomerate, and possibly two, on the top of the Biwabik iron-formation, older than the so-called Cretaceous conglomerate, beds and remnants of beds of which are found on most of the orebodies west of Eveleth. Since that time he has observed in two mines a conglomerate capping the orebody, containing large boulders which themselves were composed of the typical Cretaceous conglomerate. These conglomerates were overlain by layers of plastic shale and muds. Undoubtedly these are local lakebed or stream bed conglomerates and muds, intermediate in age between Cretaceous and Pleistocene.

Newly Discovered Fossil Remains in the Cretaceous Shale.

The conglomerate and shale found on top of many orebodies were correlated by the U. S. Geological Survey as Cretaceous from the fossil remains found in the shale, principally some teeth and vertebrae identified as probably belonging to a Mosasaur. In 1915, in the Canisteo Pit at Coleraine, one almost perfect fossil and parts of others of Ammonites were found. A reproduction is shown in Fig. 14. The specimen A, Fig. 14 is about 15 inches in diameter and 3 inches thick. Fig. B fits into the cast C. The material of which the specimens are composed is a fine iron-bearing green sand which lies on top of a coarse conglomerate about one to two feet thick. The conglomerate is unconformable on the Biwabik (Mesabi) iron formation and is composed

A

C

B

FIGURE 14—FOSSILS FOUND AT COLERAINE

principally of pebbles of iron ore with some granite and greenstone.

From the reproductions of the fossils shown herewith, Prof. W. H. Twenhofel, paleontologist at the University of Wisconsin, correlates them as probably Middle Upper Cretaceous.

Virginia Slate, Iron-Formation Contact—Because of possible value in connection with a revision of the correlation of the Huronian series in the Lake Superior district, the writer wishes to append to this paper the following record of observations of the relation of the Virginia slate to the underlying iron-formation, particularly because it is not in accord with statements as to this relation made in Monograph 52 of the U. S. Geological Survey. The latter publication states that at the top of the iron-formation and the base of the Virginia slate, there is a horizon perhaps a few hundred feet thick which is one of gradation, in which layers of iron-formation and slate alternate, and that "the layers of slate are found well down in the iron-bearing formation, and layers of the iron-bearing formation are found well up in the slate" (page 174, Monograph 52, U. S. Geological Survey).

An examination of cores from a great number of holes which penetrated through the Virginia slate into the underlying iron-formation has failed to substantiate this statement. Cores from two drill holes which penetrated between 500 and 600 ft. of Virginia slate and the entire underlying iron-formation to quartzite, and from scores of other holes, failed to show the presence of a single layer of greenalite in the true Virginia slate. Several thin layers of a carbonate rock (probably calcium carbonate) and a few crystals of iron carbonate were observed. In the iron-formation proper a very few bands of a carbonate rock were discovered. As shown on Fig. 5 of this paper, the upper horizon of the iron-formation is a slaty horizon in which layers of dark slate, chert, iron ore, and greenalite alternate. At the top of this horizon and separating it from the true Virginia slate (which is a dense dark gray or black slate), is a layer several feet thick of calcium carbonate, amorphous or very finely crystallized, together with pure chert. This layer is mentioned on page 171, Monograph 52, U. S. Geological Survey. Wherever drill holes have penetrated through the Virginia slate all along the range, this carbonate and chert layer has been found immediately beneath it. In a few holes, cores of which were examined very carefully, a small amount of conglomeratic material was found in the upper part of this layer.

Although the average thickness of the iron-formation in adjacent areas is quite uniform, as shown in the table of average dips and thicknesses, there are marked irregularities within short distances. Differences in total thickness of iron-formation of 20 to 50 ft. in drill holes ¼ mile apart are common. Two holes, 1¼ miles apart show a difference in total thickness of 121 ft.; two holes 2½ miles apart show a difference of 184 feet. Not enough close sub-division and correlation work has yet been done to determine whether such differences are due to initial deposition or erosion from the top of the iron-formation. There is so much interbedded fine conglomerate in the upper horizons of the iron-formation that we know definitely that this part of the formation at least was deposited in very shallow water. It is not at all improbable, therefore, that prior to the deposition of the Virginia slate, the iron-formation may have been raised above water, and its upper surface somewhat eroded. The variation in thickness of the Upper Slaty Horizon gives support to this idea. No marked unconformity between the two formations can be established, however. They are comformable stratigraphically, as far as now known, but the significant fact of the absolute lack of any known greenalite (so far as revealed by many years of observation of thousands of feet of drill cores by the director of explorations of the Oliver Iron Mining Company) in the Virginia slate at least for several hundred feet above the top of the iron-formation and its marked prevalence immediately beneath the Virginia slate seems to call for such a pronounced change in conditions of deposition as to demand some time interval between the two. The universal prevalence of the calcium carbonate and chert layer with some conglomerate between the two gives further support to this idea of a time interval.

These facts, minor though they may be, are presented in the hope that they may be of some value to the geologists who are engaged in the revision of the correlation of the Huronian Series of the Lake Superior district.

MINING METHOD USED AT THE BRISTOL MINE, CRYSTAL FALLS, MICH.

BY ARVID BJORK, CRYSTAL FALLS, MICH.*

At the Bristol Mine, Crystal Falls, Michigan, the mining operations may briefly be described as follows:

The shaft is located in the foot-wall about 250 ft. from the North orebody. Levels are opened up 100 ft. apart and a main cross-cut driven to the ore. This work, as well as the larger part of the development on a new level, is carried on through a winze. This winze is located in close to the ore and by doing the work in this way it does not in any way interfere with the general hoisting operations.

From the main crosscut timbered drifts are driven to the end of the orebody. These drifts are usually on the foot-wall side. Crosscuts are driven at intervals of 36 ft. and later a drift is run on the hanging side. The footwall drift is used as a main haulage drift and is driven straight while the other drift follows the rock wall.

Sub-raises, with 15-ft. pillars between, are then put up in the crosscuts. These are carried up 15 ft. over the back of the level and are not cribbed. Chutes are built in these sub-raises and, on account of the frequent blasting of chunks and hang-ups, have to be heavily constructed. The tops of these raises are then funneled out and the raises joined together and the stope opened full size from foot to hanging. The stope is then carried up in the usual back-stoping manner, breaking through to the cave above the next level. Enough of the broken ore is drawn off to allow the miners to work to the best advantage.

In places, where the orebody is over 20 ft. wide, pillars are left to help support the back. When the ore is drawn these pillars break off and crush to a great extent with their

*Superintendent of Bristol Mine, Crystal Falls, Mich.

own weight. It is necessary however to drill and blast some of the larger chunks.

Ladderways are maintained in raises carried up in the rock a safe distance back from the stope, and are connected with the stope by small drifts.

This method of mining has proven highly efficient both in regards to cost and in getting out practically all of the ore.

PROCEEDINGS OF THE TWENTY-FIRST ANNUAL MEETING, BIRMINGHAM, ALABAMA.

The Twenty-First Annual Meeting of the Lake Superior Mining Institute was arranged by the members of the Menominee Range, where the meeting would have been held had the established custom been adhered to of visiting each of the five ranges in succession. However, the members from this range advocated a trip to the iron and steel district of the South, to which the Council voted favorably, and appropriated a sum up to one thousand dollars towards defraying the expenses of the trip.

The Institute for many years assigned its annual gatherings to the iron and copper ranges of the Lake Superior district with an occasional excursion to outside points. In 1905 a trip was made to Milwaukee; in 1910 to the Indiana Steel Company at Gary and plants in Chicago; in 1914 to the city of Detroit, and in 1915 to Minneapolis. On each of these excursions the attendance was large, showing that the members enjoyed an occasional visit to other points to study the various industries closely affiliated with the mining of copper and iron ore.

Members assembled at the Congress Hotel in Chicago on Saturday, March 10th, where preliminaries were arranged, and left at 11:30 p. m. by special train, over the Queen & Crescent, for the South, and arrived in Chattanooga Sunday night. The train consisted of eight sleepers, two dining cars and a club car, and was accompanied by officials of the railway during the entire trip, and members were shown the very best attention.

MONDAY, MARCH 12TH.

After breakfasting on the train the members, including the ladies and guests, proceeded to spend the day in sight-seeing. Automobiles were on hand to take the visitors to the battlefields of Chickamauga Missionary Ridge, Lookout Mountain, and other places of interest. It is needless to state that the sight-seeing was thoroughly enjoyed, even by those who were caught in a small shower in the early part of the afternoon. Luncheon and dinner were served at the Patten Hotel. The Ishpeming members of the party, with other invited guests, were entertained at dinner by Mr. Richard Hardy and Mr. Morrow Chamberlain, former Ishpeming residents, at the Chattanooga Country Club. Later in the evening the members were entertained by the Engineers' Club, where a smoker was given. The ladies were pleasantly entertained at a theatre party.

It was in Chattanooga that the visitors from Lake Superior had the opportunity to see what damage can be caused by a real flood, and many remarked that they would rather endure the cold winters and heavy snows in preference to high water that results in so much devastation.. The Tennessee River, which flows through Chattanooga, reached the highest mark this spring, in 35 years, the record being 47 feet and 7 inches. This was about 17 feet above the average high level, and about 30 feet higher than the normal flow. Many were compelled to flee from their homes and places of business, and were just moving back about the time that the party from the North visited there. Marks left by the water could be seen on the shingles of many of the buildings, and it is remarkable that there was not a great loss of life. This was averted only by energetic work of some of the citizens as those in the low lands did not seem to realize their danger, and in many instances were removed only by force. However, the monetary loss was a heavy one, there being no insurance against floods.

After a very enjoyable day the party left Chattanooga at midnight for Birmingham.

"The name of Chattanooga is, and always will be, associated with the greatest military operations of the greatest civil war the world has ever known. Lookout Mountain, Missionary Ridge, Chickamauga and the illustrious names linked with those days of crisis all give added radiance to the fame of this noble city. And to forever perpetuate the glory of the deeds of her sons the Government has purchased for the United States all the ground of the Chickamauga battlefield and converted it into a great park, spending enormous sums annually to worthily preserve its beauty and its associations. Throughout this beautiful place are monuments erected by the different states in memory of their dead. Further, the fighting position of every battery and every division engaged has been carefully marked by guns of the pattern then in use. Numerous steel towers have also been erected, from the tops of which the whole plain, the ridge and the mountain are spread in comprehensive array. Missionary Ridge, shadowy and hazy in the distance, the dark embattlements of Lookout Mountain and the rolling green of Chickamauga, with the devoted city and the great sweeping river in the midst, make an historic and impressive picture. Lookout Mountain, the scene of the "Battle above the Clouds" and Hooker's heroic dash; Bragg's headquarters on Missionary Ridge, and General Grant's at Orchard Knob are all points of absorbing and tender interest, for, after all, it was brother fighting brother, friend facing friend; the foe was not an alien race, and every gun that was fired on either side was for the glory of the God of Righteousness. The war is past, its issues are dead, and the South is fighting a new battle, with the chances all in its favor. Its boundless resources, its glorious climate, its willing hospitality and the magnificent country of the South will not give way in this engagement; its "Bonny, blue flag" is the bright blue sky, and North and South, alike, are eager to take their stand to live forever in "Dixie." "—(Reprint.)

TUESDAY, MARCH 13TH.

Arriving in Birmingham early Tuesday morning, the party was met at the station by Messrs. Edwin Ball and C. T. Fairbairn, on whose invitation the Institute planned the trip to Birmingham. Among the members of the reception committee were several who had lived in the Lake Superior district and their meeting with old friends from the North country was very enjoyable. Headquarters were established at the Tutwiler Hotel.

The local committee had prepared a booklet entitled, "A Little Journey in the Birmingham District," a copy of which was furnished to each member of the party that they might have a comprehensive idea of the scope and importance of the resources of this section. The material compiled is very interesting and is reprinted in this volume together with the illustrations pertaining to the iron and steel interests and a map of the district showing the route traveled by the Institute.

The special train left from the Louisville & Nashville station promptly at 9:30 o'clock for the tour of inspection. The visitors were joined by the members of the local committee, who acted as guides and called attention to the many points of interest, thus adding much to the benefit and enjoyment of the trip. The order in which the mines and mills were visited is shown by the intinerary, which schedule was closely adhered to. At Bayview a real southern barbecue was served, two bands being in attendance throughout the stay.

Sight-seeing was continued after the luncheon and the party returned to the city at 6:30 p. m. Promptly at 8:30 the business meeting was held in the ball room of the Hotel Tutwiler, President Charles E. Lawrence presiding. Upon opening, Mr. Lawrence delivered an address on the "Progress of Mining in Lake Superior District," which is published elsewhere in this volume.

Telegrams and letters received by the Secretary from the following members, who were unable to attend the meeting at the last moment, were read:

Albert H. Fay, Washington, D. C.; Thomas F. Lynch, Houghton, Mich.; J. M. Longyear, Marquette, Mich.; A. J. Myers, Iron Mountain, Mich.; S. H. Pitkin, Cleveland, Ohio; G. E. Harrison, Hibbing, Minn.; Thos. F. Cole, New York City; J. M. Longyear, Jr., Marquette, Mich.; Prof. F. W. Sperr, Houghton, Mich.

The following paper was presented in oral abstract by Mr. Baxter:

*"Mining Methods on the Menominee Range"—By C. H. Baxter, Rudolph Ericson and M. E. Richards, Committee.

Under this subject is included the following list of special papers:

*"The Block-Caving System Used at the Pewabic Mine"—By A. J. Myers.

*"Methods of Mining at the Chapin Mine"—By W. C. Gordon.

*"The Method of Mining Used at the Loretto Mine"—By C. H. Baxter.

*"Mining Methods in the Iron River District of Michigan"—By Rudolph Ericson.

*"The Methods of Opening and Mining the Davidson No. 2 Mine"—By Rudolph Ericson.

*"The Sub-Stoping Method of Mining as Used at the Chatham Mine"—By F. J. Smith.

*"Mining Methods in the Crystal Falls, Amasa and Florence Districts"—By M. E. Richards.

*"Sub-Stoping in the Amasa-Porter Mine"—By M. E. Richards.

*"Mining Methods in the Florence District"—By J. M. Ridell.

*"Mining Methods Used at the Bristol Mine, Crystal Falls, Mich."—By Arvid Bjork.

This concluded the reading of papers for this session.

On motion by O. C. Davidson, the President appointed the following committee on nominations: O. C. Davidson, Iron Mountain, Mich.; F. J. Webb, Duluth, Minn.; Henry Rowe, Ironwood, Mich.; G. R. Jackson, Negaunee, Mich.; and F. H. Haller, Osceola, Mich.

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*Papers distributed in printed form.

On motion by John M. Bush, the President appointed the following committee to audit the books of the Secretary and Treasurer: John M. Bush, Ishpeming, Mich.; R. G. Whitehead, Alpha, Mich.; and W. P. Chinn, Gilbert, Minn.

On motion by J. B. Knight, the committee to be appointed on resolutions was instructed to draft a suitable resolution to Dr. Nelson P. Hulst, first President of the Institute in 1893, on the occasion of the seventy-fifth anniversary of his birthday, which occurred on February 8th, 1917.

On motion by J. R. Van Evera, the committee on resolutions was further instructed to draft a suitable resolution expressing the sentiments of the Institute in favor of universal compulsory military training and service in the Army and Navy of the United States as recommended by the General Army Board, and that the President appoint a committee of five on resolutions. The President appointed as such a committee, J. R. Van Evera, and James Russell, Marquette, Mich.; Jas. B. Knight, Norway, Mich.; Earl E. Hunner, Duluth, Minn.; and D. H. Campbell, Iron River, Mich.

All committees were directed to present their reports at the afternoon session on Wednesday.

The Secretary announced the program for the next day, whereupon the meeting stood adjourned.

WEDNESDAY, MARCH 14TH.

Promptly at 9:15 a. m. the party left the Louisville & Nashville station to continue the tour of inspection through the steel mills of the Republic Iron & Steel Company and the Tennessee Coal, Iron & Railroad Company. The train first stopped at Thomas, where the furnaces of the Republic Company are located, and then proceeded to Ensley, the largest of the plants of the Tennessee Company, where two hours were spent through the different branches of the mills. By referring to the route map the district covered by the special trains may be seen, the green lines denoting the tour made on Tuesday, and the yellow the Wednesday trip. The party returned to the city at 1:30 p. m. in time for luncheon.

At 3:00 o'clock, the members again assembled in the ball room of the Hotel Tutwiler where the business session was resumed, President Chas E. Lawrence presiding. The following papers were read by title:

*"Crushing Plant at Brier Hill Shaft"—By Floyd L. Burr.

*"Notes on the Calumet & Hecla Mine Fire"—By John Knox, Jr.

*"Mine Accidents Classified by Mining Methods for the Lake Superior District, 1915"—By A. H. Fay.

*"Herringbone Gears Used on Pumps"—By Fred M. Prescott.

*"The Founding of the Calumet & Hecla Mine, 1866-1916."

*"Electric Power in Mining on the Menominee Range"—By Charles Harger.

*"Reminiscences of the Development of the Lake Superior Iron Districts"—By J. M. Longyear.

*"Equipping and Sinking No. 1 Shaft at the Holmes Mine"—By Lucien Eaton.

"Blasting Explosives and Their Accessories"—By Charles S. Hurter.

"Record Sinking at the Homansville Shaft of the Chief Consolidated Mining Company, Tintic District, Utah"—By Walter Fitch, Jr.

"Signalling System at the Bengal Mine, Palatka, Mich."—By A. H. MacGregor.

"Stoping to Branched Raises"—By F. W. Sperr.

"The Ore Mining Method Used at the Raimund Division, Birmingham District, Alabama"—By Gerald G. Dobbs.

"Recent Geologic Development on the Mesabi Iron Range, Minnesota"—By J. F. Wolff.

"Iron Ores of Cuyuna Range"—By Harlan H. Bradt and C. W. Newton.

This concluded the reading of papers.

*Papers distributed in printed form.

REPORT OF THE COUNCIL.

Secretary's report of Receipts and Disbursements from August 31st, 1915, to March 1st, 1917:

Receipts—		Items.	Amounts.	Totals.
Cash on hand, August 31, 1915				$6,753.39
Entrance fees for 1915		$ 140.00		
Dues for 1915		2,020.00		
Back dues, 1911	$ 10.00			
Back dues, 1912	15.00			
Back dues, 1913	40.00			
Back dues, 1914	200.00			
		265.00		
Advance dues, 1916	125.00			
Advance dues, 1917	10.00			
		135.00		
Life membership		50.00		
Sale of Proceedings		133.54		
Institute Pins		4.00		
Total			$2,747.54	
Interest on deposits			303.65	
Total receipts				3,051.19
Grand total on hand and received.				$9,804.58
Disbursements—				
Stationery and printing		$ 161.60		
Postage		152.75		
Freight and express		8.62		
Exchange		.60		
Telephone and telegraphing		11.03		
Secretary's salary		1,126.15		
Stenographic work		100.00		
Editing papers		30.00		
Total office expenses			$1,590.75	
Publishing proceedings		$1,060.30		
Photographs, maps, cuts, etc		165.27		
Badges for 1915		94.38		
Donation to First-Aid contest		50.00		
Total			1,369.95	
Total disbursements				$2,960.70
*Cash on hand March 1st, 1917				$6,843.88
Grand total				$9,804.58

*Commercial bank, Iron Mountain		$1,321.76
State bank, Florence		522.12
Bonds, Town of Alpha		5,000.00
Total		$6,843.88

Report of membership at close of fiscal year:

	1916-'17. (1)	1915. (2)	1914. (3)	1913. (4)
Members in good standing	514	501	524	483
Honorary members	3	3	4	4
Life members	3	2	2	2
Members in arrears two years	43	43	19	29
Total	563	549	549	518
New members admitted	33	30	36	71

REMARKS:—

(1) Includes 66 in arrears for 1 year.
(2) Includes 77 in arrears for 1 year.
(3) Includes 54 in arrears for 1 year.
(4) Includes 34 in arrears for 1 year.
Members in arrear for 3 years and over are not included.

TREASURER'S REPORT.

Treasurer's report from August 31, 1915, to March 31, 1917:

Cash on hand, August 31, 1915	$6,753.39	
Received from Secretary	9,500.93	
Received interest on deposits	303.65	
Paid drafts issued by Secretary		$2,960.70
Cash on hand, March 31, 1917		6,843.88
Totals	$9,804.58	$9,804.58

The Auditing Committee presented the following report:

Your Committee, appointed to examine the books of the Secretary and Treasurer, beg leave to report that we have carefully examined same and find the receipts and expenditures shown therein to be in accordance with the statements of the Secretary and Treasurer for the fiscal year ending February 28th, 1917.

JNO. M. BUSH,
R. G. WHITEHEAD,
W. P. CHINN,
Committee.

On motion the report of the Committee was adopted.

REPORT OF COMMITTEE ON NOMINATION.

Your Committee on Nominations beg leave to submit the following Officers of the Institute for terms specified:

For President (one year)—Charles T. Fairbairn, Birmingham, Ala.

For Vice Presidents (two years)—Wm. D. Calverley, Houghton, Mich.; M. E. Richards, Crystal Falls, Mich.

For Managers (two years)—Thomas Hoatson, Laurium, Mich.; E. S. Grierson, Calumet, Mich.; B. W. Batchelder, Nashwauk, Minn.

For Treasurer (one year)—E. W. Hopkins, Ironwood, Mich.

For Secretary (one year)—A. J. Yungbluth, Ishpeming, Mich.

O. C DAVIDSON,
F. J. WEBB,
HENRY ROWE,
G. R. JACKSON,
F. H. HALLER,
Committee.

On motion the report of the Committee was adopted and the Secretary instructed to cast a ballot for the election of the officers for the terms specified.

The following proposals for membership are approved by the Council:

Berteling, John Francis, Superintendent, Newport Iron Mining Co., Grand Rapids, Minn.

Birkinbine, John L., Consulting Engineer and head of Birkinbine Engineering Offices, Parkway Bldg., Philadelphia, Pa.

Black, Herbert F., Executive Officer in Steel and Mining Companies, Oliver Bldg., Pittsburgh, Pa.

Botsford, Milton P., Technical Engineer, Aetna Explosives Co., 800 Torrey Bldg., Duluth, Minn.

Byington, F. J., Division Superintendent, Chicago & Northwestern Railway, Escanaba, Mich.

Clark, Harlow A., Lawyer, Harlow Block, Marquette, Mich.

Crane, Ernest Edgar, Traveling Salesman, America Manganese Steel Co., 1850 McCormick Bldg., Chicago, Ills.

Derby, Edwin Lewis, Jr., Mining Geologist, Cleveland-Cliffs Iron Co., Ishpeming, Mich.

Grigg, John, Mining Captain, Newport Mine, Ironwood, Mich.

Harger, Charles E., Superintendent of Peninsular Power Co., Iron River, Mich.

Hawes, George H., Mine Safety Engineer, 202 Torrey Bldg., Duluth, Minn.

Hughes, Charles W., Mining Captain, Amasa, Mich.

Jobe, William H., Traveling Salesman, Chicago Pneumatic Tool Co., Crystal Falls, Mich.

Kennedy, A. T., Mining Engineer, Republic Iron & Steel Co., Negaunee, Mich.

Kingston, Merton S., Consulting Engineer, Virginia, Minn.

Leonard, C. M., Superintendent of Mines, Richmond, Va.

McCallum, James G., Salesman, Keystone Lubricating Co., Houghton, Mich.

McDonald, L. N., Mining Engineer, 301 Glencoe Bldg., Duluth, Minn.

Mather, Amasa Stone, Pickands Mather & Co., Western Reserve Bldg., Cleveland, Ohio.

Morgan, Bernard A., Fuel, Lumber and Builders Supplies, Hurley, Wis.

Mitchell, Edward C., Fuse Salesman, Young Block, Houghton, Mich.

Neely, Benjamin C., Mineral and Timber Lands and Exploring, Crystal Falls, Mich.

Odgers, Ira, Diamond Drill Superintendent, Crystal Falls, Mich.

Oswald, Eugene, Superintendent, Crystal Falls, Mich.

Prescott, C. S., Mechanical Engineer, The Prescott Co., Menominee, Mich.

Rapp, Axel G. J., Engineer, Link Belt Co., Chicago, Ills..

Schaber, Carl F., Mining Engineer, Box 46, Bessemer, Ala.

Smith, F. J., Superintendent, The Brule Mining Co., Stambaugh, Mich.

St. Clair, George H., Diamond Drill Contractor, 710 Sellwood Bldg., Duluth, Minn.

Vickers, Joseph Todd, Mining Captain, Bangor Mine, Biwabik, Minn.

Whittle, Charles E., Mechanical Rubber Goods Salesman, 304 W. Randolph St., Chicago.

On motion, the Secretary was instructed to cast a ballot for the election to membership of the list as approved by the Council.

Your committee on resolutions submits the following report for the consideration of the Institute:

FIRST—Whereas, Dr. Nelson P. Hulst, the first president of this Institute, recently passed, at his home in Milwaukee, Wis., the seventy-fifth anniversary of his birth; and

Whereas, The ability, experience and courteous personality of Dr. Hulst have been of great benefit to the Institute, therefore be it resolved that the hearty congratulations and sincere well wishes of the members of the Institute, be extended to Dr. Hulst, coupled with the hope that his long and useful life may be spared many years.

And, be it further resolved that the Secretary be requested to transmit to Dr. Hulst a copy of this resolution.

SECOND—Whereas, The military history of the world, as well as the military history of our own country shows that the volunteer system of providing men for the Army and Navy is the most inefficient and most costly in life and treasure, of any known system.

And Whereas, It is the most undemocratic in its operation, because in times of necessity the patriotic citizen is compelled under this system to perform military duties in the protection and defense of our country, while the many who are fully alive to the duty they owe the country escape the performance of such duty.

And Whereas, We believe that all citizens who claim and enjoy equal rights before the Law, should be compelled to perform equal service in defense of the Law, and of our Nation.

And Whereas, Each and every head of our Army from General George Washington down to the present Chief of Staff, General Scott, have pointed out the radical defect in our volunteer system and the need of a change which will enforce the equal distribution of duties.

And Whereas, Most recent developments have clearly demonstrated that other nations do not hesitate to take advantage of a weak defensive condition, and to violate our National Rights both at home and abroad.

Now therefore, be it resolved, that The Lake Superior Mining Institute, an organization composed of more than five hundred members, in Annual Convention assembled, declare it to be their deliberate judgment that the Congress of the United States should without delay provide by Law for universal compulsory military training, and universal compulsory military service in our Army and Navy, along the line recommended by the General Army Board and that the President should sign and put into prompt execution such a law.

And be it further resolved, That the Secretary of this Institute, be and is hereby requested, to send copies of this resolution to all Senators and members of Congress representing the States of Michigan, Wisconsin, and Minnesota, and that he be further requested to give this resolution as wide publicity as possible through the press.

THIRD—Resolved, That we hereby extend our thanks to the Chattanooga Engineers Club, whose smoker was greatly enjoyed by the Institute members, and

Also to the Birmingham Committee, the Birmingham Athletic Club, and to the various companies engaged in mining and metallurgical pursuits at Birmingham and vicinity and their offcials, for the many courtesies extended and especially do we wish to express to our old friends and fellow members, Mr. Edwin Ball and Mr. C. T. Fairbairn, our keen appreciation of their efforts in our behalf.

Also to the ladies of Birmingham, who so kindly and hospitably entertained the wives of the Institute members, and

Also to the various Transportation Companies and their officials, who provided most excellent train service and extended a number of courtesies, and,

Also, to the authors who kindly submitted papers at the Institute meeting.

Also, that we extend to the President and Secretary of this Institute our sincere thanks for their untiring efforts in our interests during the past year.

All the foregoing have contributed to make this the most interesting and enjoyable meeting in the history of the Lake Superior Mining Institute.

Respectfully submitted,
J. R. VAN EVERA,
JAMES RUSSELL,
J. B. KNIGHT,
EARL E. HUNNER,
D. H. CAMPBELL.

On motion the report of the committee was unanimously adopted.

J. B. Knight called attention to mine sanitation and recommended that the subject be given careful consideration by the Institute. After some discussion the matter was referred to the committee on "Practice for the Prevention of Accidents."

The President appointed the following committee on the subject of uniformity of terms applying to different methods of mining, in accordance with the motion by Mr. Davidson at the previous session:

James E. Jopling............Marquette Range
C. H. Baxter ....Menominee Range
Frank Blackwell ....Gogebic Range
Willard Bayliss ....Mesaba Range
F. W. Sperr ....Copper Range

Announcement of the program for the next day was made, and the meeting closed.

In the evening the members were splendidly entertained by the Birmingham Athletic Club and later a lunch and smoker was tendered by the Southern Club.

THURSDAY, MARCH 15TH.

Plans for the last day of our visit to Birmingham presented a variety of attractions. Some of the members went underground in the coal mines, while others went down the iron ore mines. Many visited the National Cast Iron Pipe Company, the American Radiator Company, and other plants and also the Avondale Cotton Mills. Members of the local committee escorted the parties and furnished much information to the visitors. Automobiles were placed at the disposal of the party and trips were also made through the residence districts.

The ladies of Birmingham provided splendid entertainment for the visiting ladies throughout the stay in the city, and the experiment of having the ladies accompany the men on Institute trips proved a great success. They have only words of praise for the hospitality shown them by their Southern sisters.

The weather during the trip was very pleasant, and recollections of the cold weather and deep snows in the Lake Superior country made the contrast more noticeable and the experience more enjoyable.

Owing to the railroad strike, which was threatening the country during our visit, it was decided by the officers to cancel further trips at this time. Sincere regrets of the Institute were therefore telegraphed to the Committee at Knoxville, where the members had been invited to spend Friday, advising the necessity of a speedy return home. As the strike was called off at the last moment we regret indeed the loss of another day in the South and the pleasure of enjoying more Southern hospitality.

The following is a partial list of those in attendance:

Abbott, C. E. and wife........ .... **Bessemer, Ala.**
Backert, A. Q. and wife........ .... **Cleveland, O.**
Bandler, A. S.......**New York City**
Ball, Edwin and wife........... .... **Birmingham, Ala.**
Barber, M. H.....**Nashwauk, Minn.**
Baxter, C. H.........**Loretto, Mich.**
Berg, F. H......**Ishpeming, Mich.**
Blacklock, S. S....**Hibbing, Minn.**

Brewer, Carl ....Ishpeming, Mich.
Brigham, E. D......Chicago, Ill.
Bond, W. J.......Ironwood, Mich.
Botsford, M. P......Duluth, Minn.
Botsford, H. L.......Ontario, Can.
Brady, Samuel ..Rockland, Mich.
Bush, J. M. and wife.............................Ishpeming, Mich.
Byington, F. J....Escanaba, Mich.

Campbell, D. H. and wife....................Iron River, Mich.
Chinn, W. P. and wife...........................Gilbert, Minn.
Chisholm, A. D...Ironwood, Mich.
Clark, Harlow ..Marquette, Mich.
Close, R. C........Duluth, Minn.
Cole, W. A......Ironwood, Mich.
Collins, E. J. and wife..........................Duluth, Minn.
Cook, Chas. W..Ann Arbor, Mich.
Crane, E. E.........Chicago, Ill.

Davidson, O. C. and wife.................Iron Mountain, Mich.
Derby, E. L.....Ishpeming, Mich.

Eaton, Lucien...Ishpeming, Mich.
Erickson, C. E....Ironwood, Mich.
Ericson, Rudolph and wife...................Iron River, Mich.

Fairbairn, C. T. and wife...................Birmingham, Ala.
Fay, Joseph H..Marquette, Mich.
Fishwick, E. T....New York City
Flodin, Nels and wife........................Marquette, Mich.
Formis, Andre..Iron River, Mich.
Fraser, W. H..Crystal Falls, Mich.

Gowling, T. A. and wife.....................Marquette, Mich.
Graff, W. W.........Gwinn, Mich.
Green, A. C.........Chicago, Ill.
Grierson, E. S. and wife........................Calumet, Mich.
Grigg, John.......Ironwood, Mich.

Haller, F. H......Osceola, Mich.
Hallingby, Ole....Calumet, Mich.
Hedin, A. G.....Ironwood, Mich.
Hoatson, Thomas..Laurium, Mich.
Holley, C. E.....Bessemer, Mich.
Hughes, C. W. and wife.......................Amasa, Mich.
Hunner, E. E. and wife..........................Duluth, Minn.

Jackson, G. R....Negaunee, Mich.
Jobe, W. H. and wife...........................Palatka, Mich.

Kennedy, A. T...Negaunee, Mich.
Kingston, M. S. and wife........................Virginia, Minn.
Knight, J. B......Norway, Mich.

Lambrix, M. and wife...........................Hurley, Wis.
LaRochelle, Louis ....Houghton, Mich.
Lawrence, C. E., wife and daughter ....Palatka, Mich.
Lesselyong, Ed...Ironwood, Mich.
Letz, J. F.......Milwaukee, Wis.
Lewis, J. H.....Marquette, Mich.
Lord, E. J...Iron Mountain, Mich.
Lutes, J. S........Duluth, Minn.

Mars, W. P..........Duluth, Minn.
Matthias, A. C. and wife.........................Chicago, Ill.
Millnor, C. R. and wife.........................Chicago, Ill.
Mitchell, E. C....Houghton, Mich.
Mitchell, W. A......Chicago, Ill.
Morgan, B. A. and wife.........................Hurley, Wis.
McCallum, J. G..Houghton, Mich.
McDonald, D. B.....Duluth, Minn.
McDonald, L. N....Duluth, Minn.
McDowell, John ..Hibbing, Minn.
McGee, M. B..Crystal Falls, Mich.
MacGregor, A. H.Iron River, Mich.
McNeil, E. D......Virginia, Minn.
McNeil, E. D., Jr..Virginia, Minn.

Neely, B. C....Crystal Falls, Mich.
Newett, W. H....Ishpeming, Mich.

Odgers, Ira ..Crystal Falls, Mich.
Osborn, Chase S. and wife..... .... Sault Ste. Marie, Mich.
Oswald, E. J..Crystal Falls, Mich.

Penhallegon, W. J. and wife.... .... Birmingham, Ala.
Penton, J. A......Cleveland, Ohio
Phillips, W. G....Calumet, Mich.
Prescott, F. M..Menominee, Mich.

Raisky, F. H.......Duluth, Minn.
Rapp, A. G. J. and wife......... .... Chicago, Ill.
Reeder, J. T., wife and daughter .... Houghton, Mich.
Reigart, J. R.....Princeton, Mich.
Rice, C. W. and wife.......... .... St. Paul, Minn.
Richards, M. E. and wife...... .... Crystal Falls, Mich.
Richards, W. A................ ....Crystal Falls, Mich.
Richards, W. J................ ....Crystal Falls, Mich.
Richards, W. J..Painesdale, Mich.
Roberts, H. M................ .... Minneapolis, Minn.
Robertson, H. J..Escanaba, Mich.
Rowe, Henry....Ironwood, Mich.
Rowe, Wm. C......Duluth, Minn.
Russell, Jas. and wife.......... .... Marquette, Mich.

Schaber, C. F.....Bessemer, Ala.
Schenck, C. H.Minneapolis, Minn.
Selden, W. H., Jr..New York City
Selden, W. H., Sr.............. .... Iron River, Mich.
Sells, Max and wife............ .... Florence, Wis.
Sheldon, A. F....Marquette, Mich.
Small, H. H. and wife.......... .... Chicago, Ill.
Smith, F. G....Stambaugh, Mich.
Soady, Harry......Duluth, Minn.
Stone, W. H.........Chicago, Ill.
Stevens, Mrs. H. J............. .... Houghton, Mich.
Suess, Joseph....Negaunee, Mich.

Talboys, H. H......Duluth, Minn.
Thieman, E. A....Florence, Wis.
Thompson, H. S....Beacon, Mich.
Trebilcock, Wm.. ....North Freedom, Mich.
Trevarrow, Henry .... Negaunee, Mich.
Trevarthan, W. J.............. .... Bessemer, Mich.
Tweed, C. E........Duluth, Minn.

Van Evera, Wilbur.Virginia, Minn.
Van Evera, J. R..Marquette, Mich.

Wall, J. S. and wife.......... .... Iron River, Mich.
Warring, Frank....Florence, Wis.
Webb, F. J.........Duluth, Minn.
Wells, G. W.........Chicago, Ill.
West, W. J. and wife........... .... Hibbing, Minn.
Whitehead, R. G....Alpha, Mich.
Whittle, C. E........Chicago, Ill.
Winn, W. L......Ironwood, Mich.

Yates, W. H........Duluth, Minn.
Yungbluth, A. J..Ishpeming, Mich.

Zinn, R. P.......Ironwood, Mich.

PROGRESS OF MINING IN LAKE SUPERIOR DISTRICT 1894 TO 1917.

ADDRESS OF PRESIDENT CHAS. E. LAWRENCE, PALATKA, MICH.*

Mining—The securing of iron and copper from Nature's store-house among the rocks, located in Michigan, Wisconsin and Minnesota; contiguous to Lake Superior, continues with unabated zeal:

First—Due to rich quality.

Second—On account of large quantity.

Third—And primarily, the ratio of value received from the cost to take these ores out of the ground and the sale of them in the open market.

Comparatively, when the Institute was organized in 1894, a total of approximately 7,700,000 tons of iron ore was secured; the past year, of 1916, 66,500,000 tons, or a growth of nine times.

The tonnage of ore has varied from year to year, according to demands, yet, with this product being moved, the exploration and development has kept up the pace, and today, there is more iron ore in sight for the future than this depletion would seem to indicate, and can be duplicated many years in the future.

Copper ores secured in 1916, approximating 263,000,000 pounds at 25 cents per pound (average for last year), makes $65,750,000.

With this as a basis, which demonstrates why iron and steel remain the commercial king in the economic world, and copper a close neighbor, it is with no small amount of personal pride that we review in a resume what has brought about and kept this prestige.

Use of Tools—From the year 1870 to 1895, Lake Superior mining went through the transition of changing from hand tools to machinery in the mining of ores.

*General Superintendent, Menominee Range, Pickands Mather & Co.

From hammer, hand drill, pick and gad, with the use of lard oil and wicks for lights, we have come to use at present, air power drills, of piston type, in many forms, to suit the kind of rock or mineral worked, with acetylene and electric lights. Comparatively, the ground broken by the same gangs of men today, will show a gain of from fifty to one hundred per cent. Tram cars formerly pushed by men and mules, are now handled by motors, electric and air, in trains of five to ten cars per trip.

Machinery—Boilers and hoists have kept pace with the greater production brought to shafts; horse-power of each and higher pressures of steam serving this demand. Compressors formerly supplying forty to fifty pounds of air pressure, are not doing their duty unless double this amount is given. Electric hoists and pumps now coming into use in many places, show economy.

Steam Shovels—Steam shovels of twenty-five ton weight have been substituted by those of seventy-five to ninety tons, and lately, some of three hundred tons. Steam shovels have done more than any one thing in mine equipment, to add to the enormous tonnage handled from Lake Superior, and to give to the United States its present standing in the iron and steel world.

Shafts—These, in early days, were of shallow depth and narrow compartments, permitting only small skips.

During the past twenty-three years, we have devoted a large amount of thought to this subject, with results showing shafts in compartments—two of these for skips, hoisted and lowered in balance, of five to ten ton variation; pipe and ladder compartment; also a man and timber cage, making necessary large level openings to facilitate the easy movement of material brought through its use; this including pockets or storage bins for ore.

Formerly these were exclusively of timber, but now many are made of steel sets and cement lath, and others of cement alone, to make fire-proof, and avoid constant repair from decaying timber.

The head-frames formerly of square timber and low in height, now run to steel, varying from one hundred to one hundred fifty feet in height, permitting the hoisted material to be loaded into railroad cars.

Geology—Scientific knowledge and the Michigan School of Mines, have been the largest auxiliary factors, to guide and lead the way to success in the several departments named.

Geology is coming to be read and understood in detail, like the reading of books. Science of Force, through air, steam and electricity, are humble servants daily used and mining colleges are training the future managers on broad, practical and economical lines.

Values—The production of $200,000,000 iron ore value and $65,750,000 copper value in 1916, after fifty years of continuous operation and greater growth, tells a story, demanding the historian to aid in the recording of the details. The distribution of this fifty years annual wealth has spread to the four corners of the earth, stimulated activity wherever mining is followed; the molten metal from the ore, while changing always into products of higher value, has made the "hum of industry" and the "wheels of progress" a cheerful song of joy, calling the world to our dens and making the United States, in the short period of fifty years, a noted star in the realm of nations, due alone to the crude ore taken from Nature and changed into manufactured metallic forms of value, so that the banks are bursting with money at very low rates of interest, and the per capita is greater than at any time in history.

Man—We turn from this rosy picture to the human element entering into the product. Without a large unit of daily output, costs will go up, but with tonnage hoisted from shafts, varying from five hundred to five thousand tons per day, a basis is given around which an organization can be formed, with departments which will dove-tail into each other and give a compact whole, for low cost.

The employer, represented by foremen trained to details, watches his men and their work consistently, and guides the various movements to a completed whole. The men under his care, have come mostly from all nations in Europe—beginning in the north and extending to the south, entirely new, and have in most cases been absolute children as far as knowledge of mining is concerned. This factor has been trained and drilled to secure a product and cost that is amazing, and could only be done by wise and careful guides.

In doing of this, "Safety First" for the past five to ten years has been the prominent sign board at large installations, and this has given efficiency, or a net result of satisfaction.

Assisted in various ways, the training of men is going on continuously, through schools, day and night, through churches, moving picture shows and social relations; also trained experts sent out by the Bureau of Mines.

With this record behind to stand upon, a future lies ahead, demanding courage at the helm, and with wise, trained workers, greater prizes are sure to be ours, and these should be confined, primarily, to harmony of the workers or the human element, more than to any department, and made a special study of, to gain in the ultimate end a better citizen, which makes for a better government to the rising generation.

Nature's wealth, transferred from the ground, keeps moving in manufactured forms of details, but this in turn is making over the human being, giving a product in the living man, something always better than the previous generation.

In this alone is pleasure and joy.

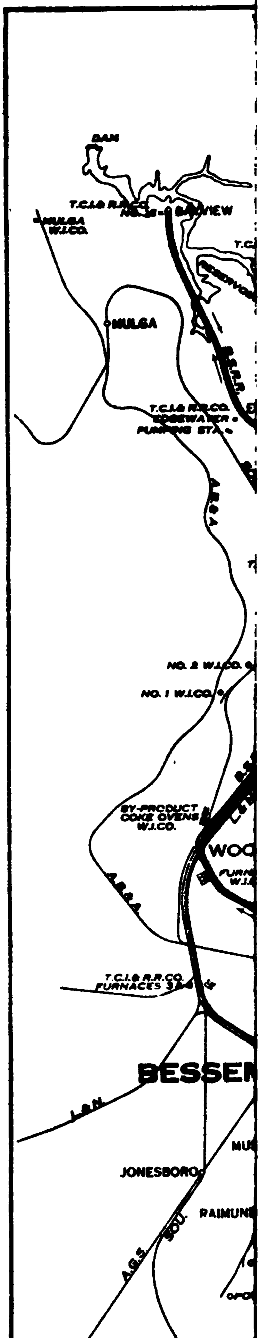
A LITTLE JOURNEY IN THE BIRMINGHAM DISTRICT.

Birmingham lies in an anticlinal valley known as the Birmingham Valley, extending seventy-five miles in length and six miles in width, embracing about 450 square miles.

Greater Birmingham includes Ensley, Pratt City, North Birmingham, East Lake, Woodlawn and East Birmingham. The 1910 census showed a population of 132,685, about one-third of which are of the negro race. The population at this time is about 200,000. The elevation as measured at the Louisville and Nashville passenger station is 630 feet. The Union Station at Pittsburgh is 743 feet, the Union Station at Chicago is 586 feet. The principal residence sections are the North Highlands and South Highlands, ranging in elevation from 700 feet to 1,000 feet. The temperature averages about 46° during the winter season, 65° for spring, about 81° for summer, and 66° for autumn. There are no large streams flowing through or near the city. An excellent quality of water is furnished by the Birmingham Water Company (not a municipal plant) which is obtained from the Cahaba river, seven miles distant from the city hall. The supply is adequate for a city of 1,000,000 inhabitants. The water is filtered, which accounts for its lack of turbidity and its bacteriological purity. Birmingham has a sanitary sewerage system which cost $500,000. One branch of the system is twelve miles long and a second branch is fourteen miles in length. Bank deposits aggregate $36,000,000, while bank clearings for the year 1916 were $145,007,387.

The city has the following transportation facilities: Louisville & Nashville, Illinois Central, Birmingham Southern, St. Louis & San Francisco, Southern Railway, Central of Georgia, Seaboard Air Line, Queen & Crescent, Mobile & Ohio, Belt Railway, Atlanta, Birmingham & Atlantic.

The tonnage of the district is more than that of the State of Georgia, and about eight times that of the Alabama cotton crop.


DAM
T.C.I.& R.R.CO.
MULGA W.I.CO.
MULGA
T.C.I.& R.R.CO.
PUMPING STA.
A.B.&A.
NO. 2 W.I.CO.
NO. 1 W.I.CO.
BY-PRODUCT COKE OVENS W.I.CO.
A.B.&A.
T.C.I.& R.R.CO.
FURNACES
L.&N.
JONESBORO
A.G.S.
SOU.

The Alabama Traction, Light & Power Company, an English corporation, has just completed a hydro-electric station at Lock 12 on the Coosa river, 65 miles from Birmingham, which will develop 100,000 h.p. The development of the power sites controlled by this company on the Coosa, Tallapoosa and Tennessee rivers, will give them a total output of about one million horsepower. Lines for the transmission of a portion of this power now reach the city.

Raw Material.

The growth and development of the district have been based on the important deposits of raw material (fuel, flux and ore).

The coal deposits occur in three distinct fields, locally designated as the Warrior-Plateau field, Cahaba and Coosa fields. The Warrior field consists of 7,845 square miles. This field carries the coking coals of the district, the most important of which are known as the Pratt and Blue Creek seams. They range from three feet to six feet in thickness. A number of partings make it necessary to wash these coals in order to obtain low ash cokes. The following are representative analyses:

	Unwashed Coal				Washed Coal			
	Ash	Vol.	Sul.	Fix. Car.	Ash	Vol.	Sul.	Fix. Car.
Pratt	8.27	30.10	1.53	61.60	5.29	30.80	1.28	63.90
Blue Creek	15.76	23.97	.86	60.27	7.93	25.61	.79	66.46

The Cahaba field includes about 150 square miles. The coals from this field are not adapted to coke making, although they possess coking qualities. They make excellent domestic, steam and gas coals. A representative lump sample has the following composition:

Ash	Volatile	Sulphur	Fixed Car.	Water.
5.59	36.79	.63	57.62	2.25

The Coosa field is small and has not been an important producer. Many of the coal seams outcrop and are operated as drift or slope mines. At other points the seams are reached by shafts. The pillar and stall method of mining is generally used. There are many installations of electric haulage and electric under-cutting machines. Shooting "off the solid" is not generally followed. Permissible explosives are largely used.

The coal production and relative standing of the chief coal-producing states as shown for the year 1916 is given below:

Rank	State	Net tons coal produced in 1916
1	Pennsylvania bituminous	175,000,000
2	West Virginia	91,000,000
3	Illinois	64 500,000
4	Ohio	37,000,000
5	Kentucky	25,300,000
6	Indiana	19,000,000
7	Alabama	16,500,000
8	Colorado	10,450,000
	December estimated.	

BIRMINGHAM COUNTRY CLUB

All of the Alabama coals above mentioned are within short distances of the consumer; in some instances the mine openings are located in the back yard of the furnace plants.

FLUX.

Limestone and dolomite are used as flux, but the latter is the more extensively used. Dolomite usually occurs in the bed of the valley, overburden being either absent or very light. Open quarry methods are followed. A representative analysis shows:

Silica (Si O2)	Alumina (Al2 O3)	Lime (Ca O)	Magnesia (Mg O)	Iron (Fe2 O3)
.55	.41	31.48	20.11	.10

Ketona and Dolcito quarries are seven miles north of the city, Limestone is mined in open quarries, most of which lie on hillsides, permitting gravity operations. A typical analysis follows:

Silica	Calcium Carbonate	Magnesia
1.64	96.60	.58

IRON ORES.

Red Ore—The red ore is in the Rockwood formation, which is probably the same as the Clinton of New York state and Pennsylvania, areas of ore of similar character being found throughout the length of the Appalachians, while the ores of Newfoundland are of almost identical character although older geologically.

The Rockwood formation comprises approximately two hundred feet of measures, principally shale and sandstone, and carries four beds of iron ore, only two of which are of commercial importance. The "Big Seam" carries the great tonnages of the district and is the only ore mined in the portion of the district covered by this trip. At Ishkooda this ore seam has an average thickness of 22 feet, the upper 11 feet of which is mined at the present time. The lower portion of the seam has a satisfactory iron content but is too high in insoluble and low in lime to be available at the present time, although from five to six feet of this lower half will eventually be mined.

This seam becomes thinner to the southwest and an average of nine and a half feet, representing the full thickness of the seam, is mined at Muscoda. The character of the ore, however, also changes to the southwest, and, while the iron content remains about the same, the ore is self-fluxing, carrying an excess of six per cent. calcium carbonate over the insoluble, while at the most northeasterly of the Ishkooda mines the insoluble is six per cent. in excess of the calcium carbonate. The change in chemical composition is gradual from the northeast to the southwest. The red ore has been worked for a distance of sixteen miles northeast of Ishkooda along Red Mountain, but the big seam becomes gradually leaner and more siliceous to the northeast, and in the northeastern half of the area the principal operations are on the Irondale seam which occurs from ten to forty feet below the big seam. The Irondale seam ranges from four to eight feet in thickness and yields an ore somewhat more siliceous than that produced from the mines on the big seam below Ishkooda.

The ore has been developed by slopes, the longest of which is at No. 8 Wenonah, where the Tennessee Coal, Iron and Railroad Company has followed the ore on the dip for a distance of 3,700 feet.

The ore, owing to leaching, was found to be high in iron and low in calcium carbonate near the surface, but this leached ore, known as "soft ore," has been practically exhausted and the unleached ore, known as "hard ore," shows no appreciable change in chemical composition with depth or distance from the surface.

The iron ore seams are sedimentary deposits laid down under water in a horizontal position and the stratification and bedding of the ore can be noted at any of the exposures along the mines. These ore beds were deposited so uniformly that the same conditions and thicknesses persist over wide areas, giving very uniform mining conditions in any one operation.

The following are typical analyses of Red Mountain ore:

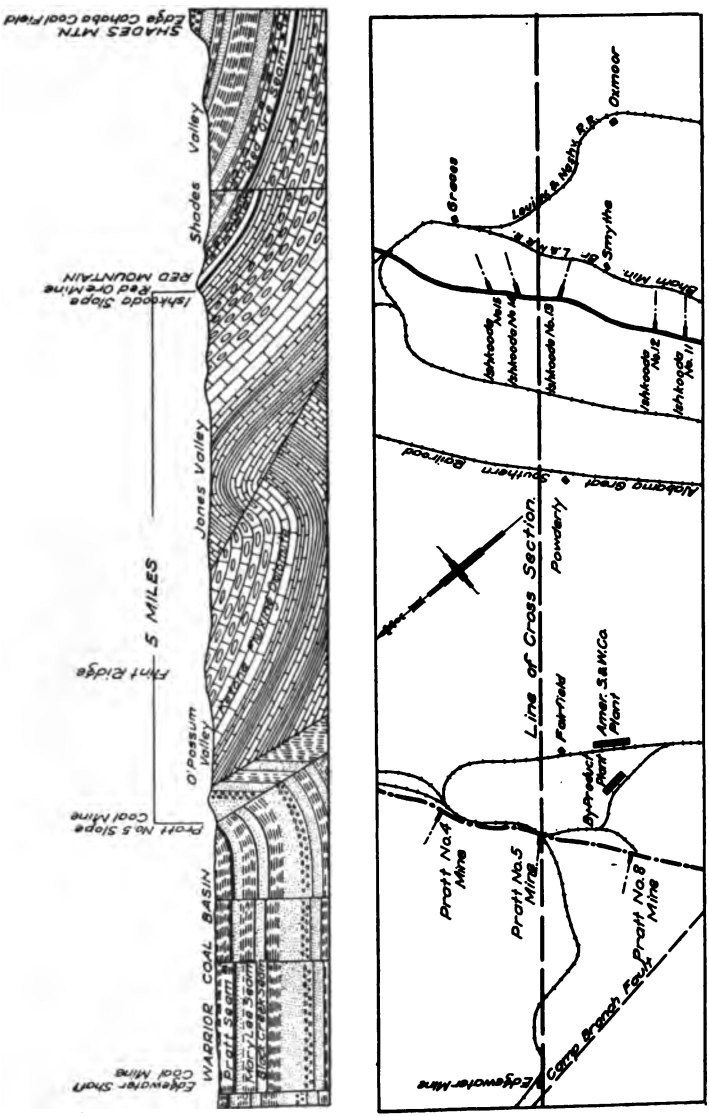
	Muscoda (No. 4)	Wenonah (No. 8)	Ishkooda (No. 13)
Iron (Fe_2O_3)	35.28	36.68	36.24
Silica (SiO_2)	10.38	13.92	16.99
Alumina (Al_2O_3)	3.12	2.96	2.98
Lime (Ca O)	19.35	16.09	14.59
Manganese (Mn)	.16	.17	.19
Phosphorus (P)	.30	.33	.35
Water	.41	.46	.44

Mining of the Red Mountain ores was first conducted on the outcrop, the overlying measures being stripped in some cases to a depth of 30 feet. This method was of short duration, the next stage consisting of slopes sunk directly in ore, approximately at right angles to the strike of the ore seam.

The merchantable ore seam averages approximately 10 feet in thickness and dips at angles varying from 12 to 45 degrees.

The older system of underground development consisted of sinking double track slopes in ore and hoisting two-ton ore cars. The Tennessee Coal, Iron & Railroad Company and Republic Iron & Steel Company abandoned this plan years ago. Slopes are now sunk ten feet below the ore seam and equipped with steel skips of twelve tons capacity.

The ore is dumped direct from two-ton steel cars at the workings into the skip. At intervals of 60 feet on the slope,



GEOLOGICAL CROSS-SECTION
OF PORTION OF
BIRMINGHAM DISTRICT
Scale: {Hor. Vert.

working headings are turned off in pairs, driven "narrow" 75 feet from the slope, and a manway raised for ventilation. The heading is then widened and driven at a width of 30 feet to the boundary. At intervals of 200 feet, raises or upsets are driven between headings to maintain ventilation. Ventilation is natural. By this method 60% of the ore is mined, 40% being left to support the overlying measures and protect the mines from inflows of water. The protective pillars will be mined when development work has ceased and will be recovered on the retreating system. Twelve-ton skips are hoisted by first motion hoisting engines which automatically dump into ore pockets at the surface, from which the ore is fed by grav-

BIRMINGHAM HIGH SCHOOL

ity into gyratory crushers, discharging direct into railroad cars.

Brown Ore—The brown ore occurs in massive and fragmentary form, associated with clay. The overburden of sand and clay varies from a very few feet to 40 feet, all of which is removed as advanced stripping, not unlike the Mesabi practice. The intermixed clay and ore is loaded by steam shovels into side dump cars delivering the pit material to the washer. The well known log washer is used for separating the ore and clay.

All massive ore is reduced by means of gyratory crushers before treatment by the washer. A representative sample of Giles pit ore carries:

Iron	Silica	Alumina	Manganese	Phosphorus	Water
50.39	13.02	2.41	.82	.32	5.38

PRODUCTION OF IRON ORE IN LEADING STATES.

Rank	State	Gross tons Produced 1915
1	Minnesota	33,464.660
2	Michigan	12,514,516
3	Alabama	5 309,354
4	Wisconsin	1,095.388
5	New York	993,845

COKE.

Coke is made in Bee Hive and Retort recovery ovens. There are 140 Koppers by-product ovens located at Woodward, which furnish the fuel supply of the Woodward Iron Company. There are 240 Semet-Solvay ovens located at the Ensley works of the Tennessee Company. This latter company has a plant of 280 ovens of the Koppers type at Fairfield, the capacity of which is about 100,000 tons of coke per month. The average yield is 71 per cent furnace coke. The by-products are: Gas, 12,800 cubic feet per ton coal; tar, 9 gallons per ton coal; ammonium sulphate, 21 pounds per ton coal.

Forty-five per cent. of the gas produced is consumed in oven operation; the surplus is pumped to the Ensley works, where it is used at the soaking pits, open-hearth furnaces, calcining plant and boilers.

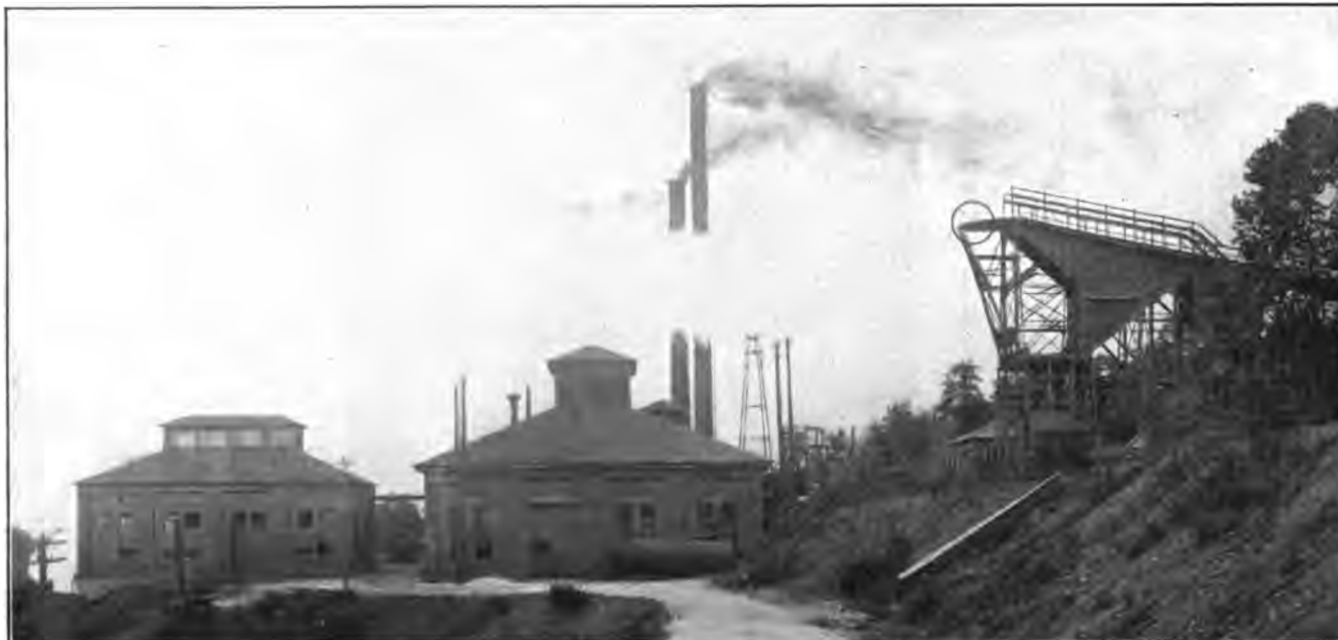
The following are representative analyses of Pratt and Blue Creek cokes:

Bee Hive Coke	Fix. C.	Ash	Sul.	Phos.	Moist.
Blue Creek	88.50	10.21	.73	.041	.70
Pratt	90.00	8.22	1.14	0.45	.61
By-Product Coke	**Fix. C.**	**Ash**	**Sul.**	**Phos.**	**Moist.**
Blue Creek	88.22	10.83	.65	.050	....
Pratt	89.98	8.88	1.16	.045	4.73

Alabama ranks second as a coke-producing state. The following statement may be of interest:

Rank	State	Net tons coke Produced 1916
1	Pennsylvania	33,388,000
2	Alabama	3,776,000
3	Indiana	3,479,000
4	Illinois	2,320,000
5	West Virginia	2,106,000

December estimated.



No. 1 Mine, Raimund—Republic Iron & Steel Co.

Republic Iron & Steel Company: Thomas, 3 stacks; coal mines; bee hive ovens; ore mines, brown and red.

American Steel & Wire Company.
United States Cast Iron Pipe and Foundry Company.
American Cast Iron Pipe Company.
Central Foundry Company.
American Radiator Company.
Bessemer Soil Pipe Company.
National Cast Iron Pipe Company.
Superior Foundry Company.

Production of Pig Iron in Leading States.

Rank	State	Gross Tons Produced 1915
1	Pennsylvania	12,790,668
2	Ohio	6,912,962
3	Illinois	2,447,220
4	New York	2,104 780
5	Alabama	2,049,453
6	Indiana and Michigan	1,986,778

Programme and Itinerary.

Tuesday, March 13, 1917.

9:15 a. m.	Lv. Birmingham (L. & N. R. R. station)	Via L. & N. R. R.
10:05 a. m.	Ar. Ishkooda (mine No. 13)	Via L. & N. R. R.
10:35 a. m.	Lv. Ishkooda	Via L. & N. R. R.
10:40 a. m.	Ar. Wenonah	Via L. & N. R. R.
10:55 a. m.	Lv. Wenonah	Via L. & N. R. R.
11:20 a. m.	Ar. Muscoda	Via L. & N. R. R.
11:50 a. m.	Lv. Muscoda	Via L. & N. R. R.
12:05 p. m.	Ar. Bessemer	Via L. & N. R. R.
12:10 p. m.	Lv. Bessemer	Via L. & N.R. R.
1:15 p. m.	Ar. Bayview	Via B. S. R. R.

Barbecue Served.

2:15 p. m.	Lv. Bayview	Via B. S. R.R.
2:25 p. m.	Ar. Edgewater (Edgewater mine)	Via B. S. R. R.
2:55 p. m.	Lv. Edgewater	Via B. S. R. R.
3:15 p. m.	Ar. Fairfield (T. C. I. by-product plant)	Via B.S. R.R.
3:40 p. m.	Lv. Fairfield (T. C. I. by-product plant)	Via B. S. R. R.
4:00 p. m.	Ar. Fairfield (Am. Steel & Wire plant)	Via B. S. R. R.
5:45 p. m.	Ar. Birmingham (L. & N. R. R. station)	Via B. S. R. R.
8:30 p. m.	Ball room, Hotel Tutwiler. Business meeting Lake Superior Mining Institute.	

Wednesday, March 14, 1917.

9:15 a. m.	Lv. Birmingham (L. & N. R. R. station)	Via B. S. R. R.
9:35 a. m.	Ar. Thomas (Republic I. & S. Co. Fces.)	Via B. S. R.R.
10:05 a. m.	Lv. Thomas	Via B. S. R.R.
10:25 a. m.	Ar. Ensley (east end furnace plant)	Via B. S. R.R.
11:10 a. m.	Lv. west end furnace plant	Via B. S. R. R.
11:20 a. m.	Ar. open hearth	Via B. S. R. R.
11:45 a. m.	Ar. rail mill finishing department	(Walk)

No. 1 Mine, Raimund—Republic Iron & Steel Co.

12:00 m.	Ar. rail mill	(Walk)
12:20 p. m.	Ar. turbo station	(Walk)
12:35 p. m.	Lv. Ensley turbo station	Via B. S. R. R.
1:10 p. m.	Ar. Birmingham (L. & N. R. R. station)..	Via B. S. R. R.
3:00 p.m	Ball room, Hotel Tutwiler. Business meeting Lake Superior Mining Institute.	
8:30 p. m.	Birmingham Athletic Club; stag entertainment.	

Thursday, March 15, 1917.

9:00 a. m. Special trips may be taken by members as follows: Guides and automobiles will be furnished.
Underground trip, Muscoda No. 4 iron mine, Tennessee Co., or Raimund No. 1, Republic Company.
Underground trip, Edgewater coal mine, Tennessee Co., or Sayreton coal mine, Republic Company.
American Cast Iron Pipe Company.
Avondale Cotton Mills.
Colgate Coal Washer, Alabama Fuel & Iron Company.
Any of these trips will prove interesting to members. A canvass will be made of all members in order that they may have the privilege of designating which trips they desire to make.

2:30 p. m. Sightseeing trips in automobiles through Birmingham residential district, or to any other places of special interest to members at their request.

Cordial invitations have been extended by the Roebuck Golf and Automobile Club, Birmingham Country Club, and Southern Club, to members to avail themselves of the privileges of these clubs and golf links during the Birmingham meeting, as follows:

ROEBUCK GOLF & AUTOMOBILE CLUB.

Birmingham, February 20, 1917.

Members of
Lake Superior Mining Institute,
Gentlemen:

In behalf of the Roebuck Golf & Automobile Club, I wish to extend to your members an invitation to avail themselves of the privileges of the Club and its golf links during their meeting on March 13, 14, and 15.

A member's badge will be sufficient identification and we hope that as many as possible will give us the pleasure of entertaining them during their stay in this city.

Yours truly,
(S.) GEO G. CRAWFORD, President.

COUNTRY CLUB.

Birmingham, February 23, 1917.

To the Members of the
Lake Superior Mining Institute,
Gentlemen:

The Country Club will be glad to have you make use of the Club and Golf Links during your meeting in Birmingham on March 13, 14, and 15.

Old Man Bogey will be on hand every day and try to make it interesting for you.

Very truly yours,
(S.) J. M. CALDWELL, President.

Blast Furnaces—Republic Iron & Steel Co.

SOUTHERN CLUB.

Birmingham, February 22, 1917.

Members of
Lake Superior Mining Institute,
Gentlemen:

On behalf of the Southern Club of Birmingham, Ala., I take pleasure in extending to you an invitation to avail yourselves of the privileges of the Club, including its cafe during your approaching meeting in this city. Your membership badge will give you entree to the Club, and we will be glad to furnish you with the regular visiting cards.

Yours very truly,
(S.) B. M. ALLEN, President.

Tuesday, March 13, 1917.

A special train will leave the Louisville & Nashville station at 9:15 o'clock Tuesday morning. Shortly after leaving the station, Alice furnace is passed on the right. It was at this plant that basic iron was first made as a market product. The annual capacity is 80,000 tons.

About four miles south of the station, the red ore mines may be noticed on the mountain side. Active operations are conducted on a line approximately twelve miles long. The ore body is continuous throughout this distance. The Tennessee Company, Woodward, Sloss, Republic, and Gulf States have mines in this territory. The large plant on the right, opposite Wenonah mine, is that of the Grasselli Chemical Company.

In ascending the mountain, the first mine reached is No. 8 Wenonah. The boiler house, hoisting engine and air compressor plant rest on the right, while on the left the store, office and camp may be observed.

Contining from Wenonah the train gradually ascends until Ishkooda No. 13 is reached, having passed in the order named.

Wenonah No. 9 mine, T. C., I. & R. R. Co.
Wenonah No. 9½ mine, T. C., I. & R. R. Co.
Wenonah No. 10 mine, T. C., I. & R. R. Co.
Songo mine, Woodward Iron Company.
Ishkooda No. 11, T. C., I. & R. R. Co.
Ishkooda No 12, T. C., I. & R. R. Co.
Clinton mine, Gulf States Steel Co.
(The outcrop of the ore may be seen here).

Ore Mine—A stop of thirty minutes will be made at No. 13, affording an opportunity of viewing the surrounding country. The elevation of the mountain at this point is 950 feet. During fair weather Birmingham may be seen lying to the northeast, Ensley stacks may be seen to the north, further to the left is the Woodward Iron Company, and, at the extreme left, are the Bessemer furnaces of the Tennessee Company.

From this point the train descends the mountain, making the next stop of fifteen minutes at Wenonah No. 8 ore mine. Leaving Wenonah, the train will proceed to Muscoda ore mines. Before reaching Muscoda, the Sloss-Sheffield Steel & Iron Company's Sloss mines will be passed on the left. A stop of thirty minutes will be made at Muscoda, where electric driven hoist and air compressors will be seen at No. 4 mine. From this point the ore mines of the Republic Iron & Steel Company at Raimund can be observed one mile to the southwest.

Barbecue—Leaving Muscoda, the train will proceed to Bayview via Bessemer, where a barbecue will be served. The interesting plants passed en route are:

Bessemer 1 and 2 furnaces, Tennessee Co.
Bessemer rolling mill, Tennessee Co.
Bessemer 3 and 4 furnaces, Tennessee Co.
Furnaces, Woodward Iron Co.
By-product and benzol plants, Woodward Iron Co.
American Steel & Wire Co.
By-product plant of the Tennessee Co.
The Harbison-Walker refractories plant.
No. 5 mine of the Tennessee Co.
Steel Cities Chemical Co., manufacturers of sulphuric acid.
Edgewater mine of the Tennessee Co.

At Bayview a slope mine is now under development. The slope is in sandstone and is inclined 26½ degrees.

Electric power will be used for all mining operations, the current coming from the Ensley works power stations.

Area of territory, 2 300 acres; average height of clean coal, 53 inches; quantity, 15,000,000 tons; proposed daily output, 2,000 tons; proposed average number of men, 750; vertical depth of coal, 240 feet; main hoisting slope "double track" 7 feet by 20 feet, 565 feet; manway 7 feet by 12 feet, 551 feet; air shaft 14 feet diameter, 240 feet deep.

Water Supply—From Bayview may be seen a portion of the impounding reservoir built by the Tennessee company during 1910 and 1911. The dam impounding this water is about 2 miles below Bayview and is about the size of the dam constructed upon Cross river of the Croton water supply for New York city.

The principal dimensions are:

	Feet.
Length at top between bastions	550
Width at base	81
Width at top	18
Height of dam	91

The water run-off area is nearly 75 square miles. The present effective capacity is 2,500,000,000 gallons of water, the submerged area is 325 acres. The capacity can be increased to 5,000,000,000 by raising the dam 15 feet. Both the up and down stream faces are constructed of concrete blocks backed with about 20 per cent. cyclopean masonry.

The Central Water Works pumping station is located at the Edgewater mine, from which it drives its fuel supply and connects with the reservoir intake by means of a straight line tunnel 6 feet 6 inches by 5 feet 6 inches in cross section extending 8,000 feet in length. This tunnel has a capacity of 75 000,0000 gallons per day.

The present pumping plant consists of two high duty cross compound Allis-Chalmers pumps, having a nominal capacity of 12,500,000 gallons each. Under service conditions the plant delivers about 270,000,000 gallons daily. A third pump of 20,000,000 gallons can be installed in this station. After leaving the pump the water flows through a 50-inch bar lock pipe 9 000 feet in length, delivering into a high level reservoir of 17,000,000 gallons capacity, from which it flows by gravity to Ensley and Fairfield. The completion of this water system made possible the recent development work of the Tennessee Company.

Coal Mine—Leaving Bayview, the next stop will be at Edgewater. At this point may be seen the surface plant of the largest mine in the state. The more important buildings are the hoisting and boiler houses, machine shop, supply house, bath house, store, and on the

OLD TANNEHILL FURNACE—REPUBLIC IRON & STEEL CO.
Operated During the Civil War on Contracts from Confederate Government

left of the railroad may be seen the school house, church and kindergarten. The central pumping station is also located here.

Coal Worked; Pratt Seam—Area of territory, 4,000 acres; average height of clean coal, 56 inches; quantity, 30,000,000 tons; proposed daily output, 3 600 tons; average number of men employed, 850; vertical depth of coal, upper landing, 238 feet; lower landing, 380 feet; bottom of shaft, 420 feet; main hoisting shaft concrete lined, airshaft, 12 feet; manway 7 feet by 12 feet, 800 feet.

By-Product Coke Plant—Leaving here the train stops 25 minutes at the by-product coke plant.

Wire Mill—The last stop is made just across the track at the new plant of the American Steel & Wire Company. This plant has daily capacity as follows: Rods, 400 gross tons; wire, 400 net tons.

Wire is manufactured into various products as follows: Galvanized wire, nails, barbed wire fencing, woven wire fencing, staples; average number of employes 1,200.

Billets for this mill are furnished by the Ensley works of the Tennessee Company, as are also electric power and water.

Wednesday, March 14, 1917.

The special train will leave the L. & N. station at 9:15 a. m. via the Birmingham Southern railroad.

Thomas Furnaces—The first stop will be made at the Thomas plant of the Republic Iron & Steel Company. There are three stacks located here in connection with the largest group of bee hive ovens in the South, comprising 910 ovens.

The company obtains an excellent dolomite from a pit quarry located near the furnace plant.

Leaving the plant of the Republic Iron & Steel Company the route is via the Birmingham Southern, passing Pratt City, at which point are located the Pratt City shops of the road, also a 2,000-ton coal washer of the Tennessee Company.

The stream over which we pass on our way from Pratt City to Ensley is the Ohio river of the Birmingham district, known locally as Village Creek.

Ensley.

At Ensley there is a group of six skip-filled blast furnaces, having an annual capacity of about 780,000 tons. During normal conditions the production is used in steel making. Two double strand Uehling casting machines permit market shipments of basic or machine cast foundry iron when necessary. Molten iron is transferred in the usual manner to the mixers, one of 250 tons capacity, and another of 600 tons.

In making steel by the Duplex process, which is the method used at Ensley, the iron is blown in 20-ton converters. The converter performs the usual Bessemer function of eliminating partially or completely the silicon, manganese and carbon, phosphorus and sulphur being later removed in the open hearth. The iron is either full blown, in which all of the silicon, manganese and carbon are eliminated, or "high blown," in which the silicon and manganese have been eliminated but a portion of the carbon remains.

The total carbon carried in the iron is about 3.8 per cent. It is practicable to get within 25 per cent. of the carbon specified. The following table is illustrative of the Bessemer practice:

MIXER METAL.

C.	Si.	S.	Phos.	Mn.
3.8	.91	.041	.81	.40

Blown Metal.

Ladle	Si.	C.	Mn.
1	.01	.07	.01
2	.02	.06	.02
3	.02	.06	.02
4	.03	2.76	.05
5	.03	3.10	.05

The blown metal is transferred to the open-hearth furnaces, each of which is of 100 tons capacity. They measure 45 feet from port to port and are 16 feet wide. Before introducing the blown metal they are charged with cold stock, consisting of calcined lime, ore, scale and scrap in the order named.

When making rail steel it is the practice to charge three "soft" or "full blown" converter heats and two "high blown" heats, the latter containing about 3.00 per cent carbon. A rather violent re-action or "kick" characterizes the addition of the first high carbon heat; the second addition produces a milder re-action. After the bath has again become normal, tests are taken for phosphorus and carbon, and if these are satisfactory, the full heat is quickly poured into a 100-ton ladle. Re-carburizing is effected by ladle additions of coke.

The open-hearth plant has an annual capacity of 950,000 tons.

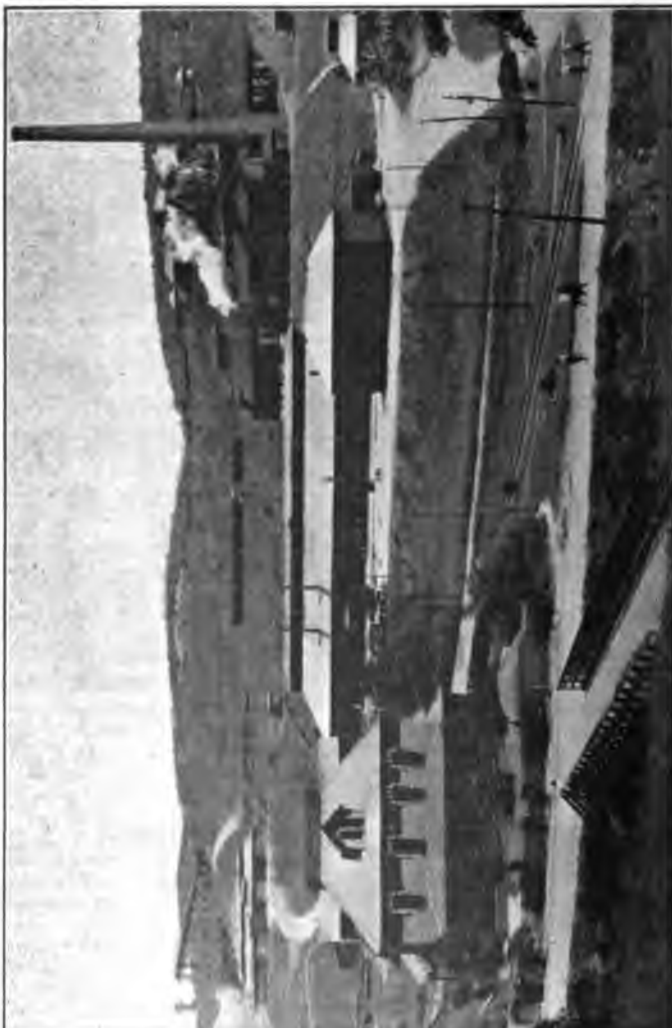
The 44 inch blooming mill is driven by 55 inch by 66 inch direct-connected Mesta reversing engine. Ingots are 24 inch by 24 inch in section at the base, and average about 9,500 pounds in weight. This mill furnishes 8 inch by 8 inch blooms for the rail mill and also for the billet mill. The blooming mill has produced 62,000 tons as a monthly record, while the record of the rail mill is 46,000 tons, which can be readily increased by additional finishing capacity.

The steam requirements have been so largely reduced as a result of installing mixed pressure turbo blowers at the furnaces and turbo generators at the rolling mills, that under normal conditions the surplus steam generated at the blast furnaces is sufficient to operate the mills. A 16-inch steam line 3,200 feet long was installed to carry the surplus available and a 14-inch line has since been added.

Leaving Ensley, the party will return via the Birmingham Southern, detraining at the Louisville & Nashville station.

Historical Data.

1818 First blace furnace in Alabama (Russellville).
1827 First coal mined in Warrior coal field.
1854 First coke made from Alabama coal (foundry use).
1858 First rolling mill built (Shelby).
1863 Oxmoor furnace built (Birmingham district).
1864 Red hematite from Red Mountain first used.
1871 Birmingham founded.
1878 First coke pig iron made in Alabama (Oxmoor).
1879 First rolling mill built in Birmingham.
1888 First steel made in Alabama was poured March 8th at the plant of the Henderson Steel Manufacturing Company, North Birmingham.
1895 Basic iron first made at Alice furnace, July 12.
1899 The first steel made by the Tennessee Coal, Iron & Railroad Company was poured Thanksgiving Day, November 30th. This plant consisted of ten 50-ton tilting furnaces of the Wellman type. These furnaces were abandoned in 1908 and replaced by the present 100-ton furnaces.



Nos. 4 and 5 Mines, Muscoda—Tennessee Coal, Iron & Railroad Company

CHATTANOOGA.

Monday, March 12, 1917.

6:30 to 8:30 a. m.	Breakfast on dining car.
8:45 a. m.	Leave Chattanooga Terminal station in sight-seeing cars for trip through Chickamauga National Military Park, Missionary Ridge and the National Cemetery. Stops will be made at all prominent monuments.
12:00 m.	Luncheon will be served at Patten Hotel.
2:00 p. m.	Leave Patten Hotel for the trip to the summit of Lookout Mountain in sight-seeing cars.
5:00 p. m.	Dinner.

KNOXVILLE.

Friday, March 16, 1917.

7:00 a. m.	Arrive Knoxville. Breakfast on dining car. The day will be spent in visiting points of interest at Knoxville and Mascot.
5:00 p. m.	Leave Mascot.

BIRMINGHAM LOCAL COMMITTEE

Abbott, C. E.
Badham, Henry
Ball, Edwin
Ball, E. M.
Banister, R. H.
Barnes, E. M.
Beecher, L. T.
Bowron, James
Brooks, T. E.
Brown, J. R.
Bush, Morris W.
Carey, A. W.
Coffin, Harry
Crawford, Geo. G.
Crawford, Webb
Crockard, F. H.
Cross, C. B.
Cutler, F. G.
Davidson, Jas. L.
DeBardeleben, Henry
DeBardeleben, C. F.
Dobbs, G. G.
Ellis, E. E.
Estep, F. L.
Fairbairn, C. T.
Faull, R.
Fear, T. G.
Fletcher, John F.
Friese, J. E.
Geismer, H. S.
Hamilton, Robert
Harris, J. R.
Hassinger, W. H.
Hillman, Gentry
Kestler, Fred
Kirkpatrick, J. D.
Landgrebe, Karl
Lowman, Chas.
Lide, M. J.
McCormack, G. B.
McHugh, J. M.
McQueen, J. W.
McWane, J. R.
Maben J. C.
Mathias, W. G.
Meagher, J. F.
Moffet, C. W.
Mitchell, W. E.
Morris, F. G.
Moulton, W. C.
Noland, Lloyd
Owens, D. W.
Penhallegon, Will J.
Ramsey, Erskine
Roberts, Dave
Ryding, H. C.
Salmon, H. S.
Schaber, C. F.
Seibels, H. G.
Sevier, L.
Shook, Warner
Shook, Pascal
Smith, H. S.
Sterling W. H.
Swann, Theodore
Temple, T. H.
Urban, H. M.
Wells, Oscar
Wilson, Willard
Winslow, F. B.
Woodward, A. H.

REMINISCENCES OF THE UPPER PENINSULA OF MICHIGAN.*

BY F. W. HYDE, CLINTONVILLE, WIS.

The following interesting article relative to the early history of the upper peninsula is from the pen of F. W. Hyde, who is now engaged in the real estate business at Clintonville, Wisconsin:

In the Iron Mountain Press of March 15th there was a fine article on the early reminiscences of the Menominee iron range by the Honorable John Longyear. It may be of interest to some of the many readers of your valuable paper to have recalled some of the early history of the northern peninsula, as remembered by the writer who first visited that section in 1866, coming from Green Bay to Escanaba by boat, there being no railroad between these two places at this early time.

Escanaba was a small village, called Sand Point. There were only a few business places, two hotels, the Tilden House, owned by the railroad, and the Oliver House, owned by a Mr. Oliver. One large ore dock and a large commercial dock owned by the railroad company.

The population was mostly railroad men and lumbermen. Ex-United States Senator Isaac Stephenson, of Marinette, and associates, owned a small water mill at the mouth of the Ford and one at the mouth of the Cedar rivers. There was some lumbering done on the White Fish river, and as we now remember, a saw mill operating there.

The Chicago & Northwestern railroad had built from Escanaba to Negaunee for the purpose of hauling ore from the mines to Escanaba to be shipped to lower ports, connecting at Negaunee with the Marquette & Ontonagon railroad for Marquette.

Leaving the railroad at Shake town, now called Swanzey, we traveled west across the east part of the Escanaba and

*From Iron Mountain Press, May 3rd, 1917.

the main Escanaba river until we reached the tributaries of the Ford river. Here we located some of the finest of pine timber. The most of this whole country was government and state lands, there being very little entered. That winter the Isaac Stephenson company lumbered on the east part branch of the Escanaba as far north as township 45, range 25, due west from Swanzey.

To show the difference in lumbering in those early days and the present time, that winter the company hauled logs on long sleds—that is, sleds with runners twelve or fourteen feet long—without any iron shoes on them. They were called wooden-shod sleds. One mile was considered a long haul. The camps were very primitive, built low with two or three small windows, a fire place located in the center, over which the cooking was done, baking their bread or biscuit in a tin baker, their beans in an iron kettle, by digging a hole in front of the fireplace, placing in the kettle and covering it over with hot coals and ashes. This was called the "bean hole." Their principal living was bread, potatoes, beans, fat pork, dried apples, tea and black strap molasses.

We remained in the woods until about the middle of December. Then came to Shake town again.

In the summer of 1867 we went to Lake Michigamme and down the Michigamme river to the mouth of the Deer river, in town 44, range 32. The Portage Lake Canal company had commenced making selections of the lands ceded to them by the government for the building of a canal across Keweenaw Point. Here at the mouth of the Deer river we met Albert Buybec and men who had preceded us by a few days, and were looking lands for the canal company.

These lands had been withdrawn from the market until such time as the canal company could make their selections.

After spending several weeks looking timber, we concluded to go out of the woods. Reaching the mouth of the Deer river, we again met Buybec and men, also William Culbertson and Sam Short, both from Girard, Penn. Mr. Culbertson, later on, became heavily interested in lumbering on the Menominee and tributaries. After talking the matter over, we concluded to join forces and travel out through the woods to the railroad instead of going up the river by boat.

After four days of hard travel we reached Shake town, going from there to Negaunee and Marquette.

Again, that fall, early in November, we went to Champion, which had started that summer, this being the terminus of the Marquette & Ontonagon railroad. The mining company had just opened the Champion mine and was building a large blast furnace for smelting the ore. Mr. Donkersley, of Marquette, was the president of the road and was also heavily interested in the blast furnace. There was a small log-house used for a store, several small log-houses for living purposes and a small log-house kept by a Frenchman as a boarding-house. A Mr. Case, a very nice and refined gentlemen, was general manager. A Mr. Wilson was captain of the mine, and a Mr. Doty store-keeper.

We remained here for two or three days and built two small flat-bottomed boats to use in going down the Michigamme river.

Purchasing our supplies here we floated down the Michigamme lake to the outlet, then down the Michigamme river to the mouth of the Deer river, where we made our headquarters.

There were no signs of civilization after leaving Champion. The Republic did not start for some time after this.

Leaving our supplies, such as we could not carry, at the mouth of the Deer river, covered with our boats, we looked timber between the Deer river and the Paint. There were large bodies of pine, hemlock, hardwood and cedar, although we only located the very choicest of pine timber, there being no sale for any other kinds of timber at this early time. There was an abundance of fur-bearing animals of all kinds, as the country had never been hunted or trapped, except what little had been done by the Indians.

After remaining in the woods until Christmas, and being afraid of the deep snows coming on, we decided to start for civilization. Taking a southerly course through the woods, we reached the Menominee river, a few miles north of Bad Water, an Indian village, where there were between one and two hundred Chippewa Indians living. It being a bitter cold day, we stopped at one of the wigwams to get warm and see if we could get something to eat. Having a small allowance of flour with us, we made the squaw understand we were hungry and wished her to bake us some bread, which she very readily did, also cutting a nice sirloin of venison and cooking it with boiled potatoes and onions. She soon placed before us a dinner that would do justice to the most expert feminine cook. Leaving Bad Water we took the Indian trail that lead down to Sturgeon falls. That night we camped on the pine plains near Lake Antoine.

The next morning, after crossing the lake, we missed the trail, and for this reason again took our course through the heavy forests, crossing over the high ridge where the City of Iron Mountain is now located. After leaving the plains south of Lake Antoine the country became rolling and was covered with heavy growth of white and Norway pine, with very little underbrush. There were thousands of deer and this whole section of country resembled a sheep ranch. Late in the afternoon we reached the Menominee river at Little Quinnesec Falls. Traveling down the river on the ice until long after dark we came to an Indian camp. Obtaining permission, we stayed with him over night. The Indian who could talk some English, told us there was a lumber camp just below the Sturgeon Falls, which was about two miles down the river. This was the fartherest up the lumbering had been. Leaving the Indian early the next morning we soon reached the lumber camp, where we had breakfast. From there we were three days reaching Menominee.

In the spring of 1868, the writer in company with a man named Rolin Crane, who made his headquarters at Menominee, and who had looked timber with George Kitson, a half-breed Indian, left Menominee and traveled to Lake Antoine. There were no habitations of any kind only what were called stations, or stopping places, the first one being twelve miles from Menominee kept by a Mr. Drimic. The next place, twenty-five miles from Menominee, called Grand Rapids, was kept by a widow lady named Mrs. McFaden. We reached here a little before sundown the first day. Soon after we reached here, Bartley Breen came in from the woods, and walked to Menominee that night. He had located the Breen mine and was on his way to Marinette to enter it.

At the mouth of the Sturgeon river we found a man, who was a product of the Emerald Isle, living with a squaw. He had a small clearing started, and a small log house in which he was living. This later on became what was known as the New York Farm.

In the summer of 1869, we located timber on the Sturgeon river, a branch of the Ford, and the main Ford river. Then crossed over to the Michigamme. We left Escanaba in August, and remained in the woods until about the middle of December, coming out at Champion. When we reached the mouth of the east fork of the Ford river, which is in town 43, range 29, we found a log camp that had been built by John Armstrong and Joe Gaushey in the fall of 1866. This was the only habitation or sign of civilization in this whole country. This camp was used as their headquarters for trapping. John Armstrong was from Negaunee, where he had been employed by Edward Breitung as head bookkeeper.

The first day of April, 1870, we went from Negaunee to the foot of Lake Michigamme with a team. There was over four feet of snow on the level. The railroad had discontinued running trains between Marquette and Champion on account of the deep snows and small amount of business. Ishpeming had just started, there was a number of small log houses built on what is now the main street. This road was built from Marquette to L'Anse by Timothy Hurley, of Marquette, who ran a line of stages during the winter time between these two places.

We traveled from the foot of Lake Michigamme to the mouth of the Deer river on snow shoes, where we had our supplies cached. We looked timber on the head of the Fence and Deer rivers. Then crossed over the Ford and looked timber in town 44 and 45, range 28 and 29, and along the headwaters of the Sturgeon, coming out of the woods at Escanaba.

In August, 1872, we left Escanaba in a sail boat for the White Fish river. We looked timber along this river as far north as Trout Lake. Also on the Sturgeon river that empties into the Big Bay du Noc. At this time there were large tracts of the very finest of hard maple and some of the finest cedar to be found in the whole country. We returned to Escanaba in October. The railroad company had commenced building the road from Escanaba west and south towards Menominee, and were running a construction train out of Escanaba for eight or ten miles. Obtaining permission from the superintendent, we rode out four miles to what was afterwards called Pine Ridge. Here we selected the southwest quarter of section 16, town 39, range 23. This proved to be one of the best quarter sections we ever located, it being large primitive cork pine that would average about two and one-half logs to the thousand feet. We looked timber along the railroad to the crossing of the Little Cedar river. It being late in December, we concluded we had enough of woods work and started for Menominee. The construction train ran out of Menominee north for about fifteen miles. Obtaining permission from the conductor, we road to Marinette. That night we stayed at the Bagley House, which was kept by Joe Jeramy, Sr., a Frenchman. The hotel being crowded to its utmost and due to our personal appearance after being in the woods for nearly four months, we were not acceptable as guests of a first-class hotel. After some persuasion we were allowed the privilege of sleepng on the soft side of a hardwood floor for the small sum of fifty cents.

That winter, or early the next spring, the railroad company cut out the right-of-way from 42, now called Spaulding, for seven or eight miles toward the Breen mine. This was called the Breen mine until the railroad was built, and then was changed to Waucedah.

In June, 1873, we traveled from Spaulding over this route to section 32, town 40, range 30. There was a log house built at the Breen mine and some explorations had been started. A small clearing had been made, and a small patch of potatoes and other vegetables planted. For the first seven or eight miles from 42 towards the Breen mine it was through a thick cedar and tamarack swamp. After reaching the end of where it was cut out, we followed a blazed trail until we came to the Breen mine. From here we followed the trail until we crossed the Sturgeon river, following down the river, over an old lumber road, until we reached the New York Farm, at the mouth of the Sturgeon. A Mr. Rice was in charge and was general manager. This was headquarters for the New York Lumber company. Here they raised a large amount of potatoes, cut hay, and pastured their teams during the summer season.

Following up the Menominee river over a narrow woods road to the Little Quinnesec Falls, we reached a log camp owned by Messrs. Davis and Smith, of Oshkosh, who had done some lumbering here the previous winter. The next place was at the end road, on section 34, town 40, range 30. Here was a small log camp owned by William and James Dickey, who kept a small supply of provisions, and were trading with the Indians. This was the only place between here and Menominee a distance of seventy-five miles over a woods road, that anything in the line of provisions could be obtained.

Early that spring, Welcome Hyde, of Appleton, commenced exploring for iron on section 22, town 40, range 30, near Lake Fuma. Some of the land had been burnt over and they cleared off a small piece and planted potatoes and other vegetables, which came in very handy later on. Soon after this they moved to section 22, town 40, range 30, where they did some exploratory work. The Menominee Iron company commenced exploration that spring, on section 16, town 39, range 39. N. P. Hulst was general manager and Mr. Whitehead captain of the mine.

The Hon. John Buell commenced explorations in the southwest part of town 39, range 30 and as we remember now, on section 32. In July he struck a large vein of blue hematite ore. This was the first ore discovered of any amount by digging test pits. There were various outcroppings in different places, the largest one being on section 31, town 40, range 30, now called the Millie mine. In the latter part of September, we built a small log house near the west line of section 32, now called the Pewabic mine, and cut out a road to Dickey's trading post, a distance of two miles. This was the first house of any kind built at Iron Mountain.

At that time it seemed as though we were at quite a high elevation, as we could look over the top of the tall pine timber below us for miles, and could see over into Wisconsin. Even at this high elevation we dug for water, and had dug only a few feet when we struck a vein of pure cold water. As the land was a sandy soil, in order to keep it from caving in, we split short thin pieces of timber and drove them down around the outside of the spring, the water raising and flowing over near the top. After we completed our house and moved in, we decided to celebrate the occasion by having a feast. One or two days before the alloted time, one of the men, who claimed to be an expert with the gun, went out and in a short time killed a nice large fat deer. Preparing it for cooking with some of our new potatoes and other vegetables we had raised, we soon had a feast that would satisfy the most exacting epicure, and though our dress suits, and champagne were not in range, our appetites were, and we did ample justice to all the eatables placed before us.

Our first visitors were the late United States Senator Sawyer, from Oshkosh, and Jud Hayward, from Shawano, Wisconsin. Soon after this we were favored by a visit from Professor Pompelli and party. The next spring, after the snow had disappeared, we were visited by several parties who were looking for timber or iron. H. D. Fisher and Frank Keyes, from Menasha, Wisconsin, were callers, on their way to where the City of Florence is now located. Mr. Fisher discovered the Florence mine the previous year.

Soon after this we suspended operations and bid adieu to this part of the country, and did not have the pleasure of visiting it again for twelve or thirteen years. Iron Mountain during this interval had started, and become quite a city.

On our last visit there we walked up to where our log house was built and found it occupied by the mining company. The water was flowing from our spring just as we had left it unmindful of the many changes that had taken place during these long years. And just for the sake of good luck, we once more took a drink from its delicious waters.

PAST OFFICERS.

PRESIDENTS.

Nelson P. Hulst	1893	James MacNaughton	1905
J. Parke Channing	1894	Thomas F. Cole	1906
John Duncan	1895	Murray M. Duncan	1908
William G. Mather	1896	D. E. Sutherland	1909
William Kelly	1898	William J. Richards	1910
Graham Pope	1900	F. W. Denton	1911
W. J. Olcott	1901	Pentecost Mitchell	1912
Walter Fitch	1902	W. H. Johnston	1913
George H. Abeel	1903	L. M. Hardenburgh	1914
O. C. Davidson	1904	C. E. Lawrence	1915

(No meetings were held in 1897, 1899, 1907, and 1916).

VICE PRESIDENTS.

1893.

John T. Jones		Graham Pope
F. P. Mills	J. Parke Channing	M. W. Burt

1894.

John T. Jones		Graham Pope
F. P. Mills	R. A. Parker	W. J. Olcott

1895.

F. McM. Stanton		Per Larsson
Geo. A. Newett	R. A. Parker	W. J. Olcott

1896.

F. McM. Stanton		Per Larsson
Geo. A. Newett	J. F. Armstrong	Geo. H. Abeel

1898.

E. F. Brown		Walter Fitch
James B. Cooper	Ed. Ball	Geo. H. Abeel

1900.

O. C. Davidson		J. H. McLean
T. F. Cole	M. M. Duncan	F. W. Denton

1901.

J. H. McLean		F. W. Denton
M. M. Duncan	Nelson P. Hulst	William Kelly

	1902.	
William Kelly Nelson P. Hulst	Fred Smith	H. F. Ellard Wm. H. Johnston
	1903.	
H. F. Ellard Fred Smith	James B. Cooper	Wm. H. Johnston John H. McLean
	1904.	
H. F. Ellard Wm. H. Johnston	Fred Smith	John H. McLean James B. Cooper
	1905.	
M. M. Duncan Fred M. Prescott	F. W. McNair	John H. McLean James B. Cooper
	1906.	
M. M. Duncan J. M. Longyear	Fred M. Prescott	F. W. McNair F. W. Denton
	1908.	
J. M. Longyear F. W. Denton	David T. Morgan	D. E. Sutherland Norman W. Haire
	1909.	
W. J. Richards Charles Trezona	D. T. Morgan	D. E. Sutherland Norman W. Haire
	1910.	
W. J. Richards John M. Bush	Frederick W. Sperr	Charles Trezona James H. Rough
	1911.	
E. D. Brigham John M. Bush	Frederick W. Sperr	C. H. Munger James H. Rough
	1912.	
E. D. Brigham Geo. H. Abeel	W. P. Chinn	C. H. Munger W. H. Jobe
	1913.	
Geo. H. Abeel Francis J. Webb	W. P. Chinn	A. D. Edwards W. H. Jobe
	1914.	
Francis J. Webb Charles T. Kruse	Luther C. Brewer	A. D. Edwards Charles E. Laurence
	1915.	
C. T. Kruse C. E. Lawrence	G. R. Jackson	L. C. Brewer T. A. Flannigan

MANAGERS.

	1893.	
John Duncan Walter Fitch	William Kelly	James MacNaughton Charles Munger

	1894.	
Walter Fitch John Duncan	M. E. Wadsworth	C. M. Boss O. C. Davidson
	1895.	
F. P. Mills Ed Ball	M. E. Wadsworth	C. M. Boss O. C. Davidson
	1896.	
F. P. Mills Ed. Ball	C. H. Munger	Graham Pope William Kelly
	1898.	
M. M. Duncan J. D. Gilchrist	T. F. Cole	Graham Pope O. C. Davidson
	1900.	
E. F. Brown Ed. Ball	James B. Cooper	Walter Fitch George H. Abeel
	1901.	
James B. Cooper James MacNaughton	(One Vacancy)	James Clancey J. L. Greatsinger
	1902.	
James Clancey J. L. Greatsinger	Amos Shephard	Graham Pope T. F. Cole
	1903.	
Graham Pope Amos Shephard	W. J. Richards	T. F. Cole John McDowell
	1904.	
John McDowell Wm. J. Richards	Graham Pope	Thomas F. Cole Amos Shephard
	1905.	
John C. Greenway John McDowell	William Kelly	H. B. Sturtevant Wm. J. Richards
	1906.	
John C. Greenway Jas. R. Thompson	William Kelly	H. B. Sturtevant Felix A. Vogel
	1908.	
James R. Thompson Felix A. Vogel	John C. Greenway	J. Ward Amberg Pentecost Mitchell
	1909.	
F. E. Keese W. J. Uren	L. M. Hardenburg	J. Ward Amberg Pentecost Mitchell
	1910.	
Frank E. Keese Charles E. Lawrence	William J. Uren	L. M. Hardenburg William J. West

1911.

Charles E. Lawrence		William J. West
Peter W. Pascoe	J. B. Cooper	L. C. Brewer

1912.

M. H. Godfrey		J. E. Jopling
Peter W. Pascoe	J. B. Cooper	L. C. Brewer

1913.

M. H. Godfrey		J. E. Jopling
G. S. Barber	Wm. H. Johnston	C. H. Baxter

1914.

G. S. Barber		C. H. Baxter
W. A. Siebenthal	*Stuart R. Elliott	J. S. Lutes

*To fill vacancy of Wm. H. Johnston, elected to presidency.

1915.

W. A. Siebenthal		J. S. Lutes
Henry Rowe	M. E. Richards	Enoh Henderson

TREASURERS.

C. M. Boss 1893
A. C. Lane 1894
Geo. D. Swift 1895-1896
A. J. Yungbluth 1898-1900
Geo. H. Abeel 1901-1902
E. W. Hopkins ....1903-....

SECRETARIES.

F. W. Denton 1893-1896
F. W. Denton and F. W. Sperr....................................1898
F. W. Sperr 1900
A. J. Yungbluth ....1901-....

LIST OF PUBLICATIONS RECEIVED BY THE INSTITUTE.

American Institute of Mining Engineers, 29 West 39th Street, New York City.

Mining and Metallurgical Society of America, 505 Pearl Street, New York City.

American Society of Civil Engineers, 220 West 57th Street, New York City.

Massachusetts Institute of Technology, Boston, Mass.

Western Society of Engineers, 1734-41 Monadnock Block, Chicago.

The Mining Society of Nova Scotia, Halifax, N. S.

Canadian Mining Institute, Rooms 3 and 4, Windsor Hotel, Montreal.

Canadian Society of Civil Engineers, Montreal.

Institute of Mining Engineers, Neville Hall, Newcastle Upon-Tyne, England.

North of England Institute of Mining and Mechanical Engineers, Newcastle-Upon-Tyne, England.

Chemical, Metallurgical and Mining Society of South Africa, Johannesburg, S. A.

American Mining Congress, Munsey Bldg., Washington, D. C.

State Bureau of Mines, Colorado, Denver, Colo.

Reports of the United States Geological Survey, Washington, D. C.

Geological Survey of Ohio State University, Columbus, O.

Geological Survey of New South Wales, Sydney, N. S. W.

Oklahoma Geological Survey, Norman, Okla.

University of Oregon, Library, Eugene, Oregon.

Case School of Applied Science, Department of Mining & Metallurgy, Cleveland, Ohio.

University of Illinois, Exchange Department, Urbana, Ills.

University of Missouri, Columbia, Mo.

University of Michigan, Ann Arbor, Mich.

University of Colorado, Boulder, Colo.

Columbia University, New York City, N. Y.

University of Pittsburg, State Hall, Pittsburg, Pa.

Iowa State College, Ames, Iowa.

Iron Age, 239 West 39th Street, New York.

Engineering & Mining Journal, 10th Avenue and 36th Street, New York.

Engineering Magazine, 140 Nassau Street, New York.

The Mining Magazine, 724 Salisbury House, London, E. C.

Mines and Mining, 1824 Curtis Street, Denver, Colo.

Engineering-Contracting, 355 Dearborn Street, Chicago, Ills.

Mining Science, Denver Colo.

Mining & Scientific Press, 420 Market St., San Francisco, Cal.

The Mexican Mining Journal, Mexico City, Mexico.

Stahl und Eisen, Dusseldorf, Germany, Jacobistrasse 5.

The Excavating Engineer, 267 National Avenue, Milwaukee, Wis.

LAKE SUPERIOR IRON ORE SHIPMENTS FROM THE DIFFERENT RANGES FOR YEARS PRIOR TO 1915, 1915, 1916, AND GRAND TOTAL FROM 1855 TO 1916, INCLUSIVE.

		Prior to 1915.	1915.	1916.	Grand Tot.
Marquette Range	(Tons	109,709,669	4,105,378	5,396,007	119,211,054
	(Per cent	16.6	8.7	8.1	15.5
Menominee Range	(Tons	92,260,269	4,982,626	6,364,363	103,607,258
	(Per cent	14.1	10.6	9.5	13.4
Vermilion Range	(Tons	35,846,066	1,733,595	1,947,200	39,526,861
	(Per cent	5.5	3.6	2.9	5.1
Gogebic Range	(Tons	80,845,441	5,477,767	8,489,685	94,812,893
	(Per cent	12.3	11.5	12.7	12.3
Mesabi Range	(Tons	334,571,935	29,756,689	42,525,612	406,854,236
		51.0	63.0	63.9	52.9
Cuyuna Range	(Tons	2,044,967	1,136,113	1,716,218	4,897,298
	(Per cent	.2	2.4	2.6	.6
Miscellaneous	(Tons	1,442,744	80,583	219,381	1,742,708
	(Per cent	.2	.2	.3	.2
Total tons		656,721,091	47,272,751	66,658,466	770,652,308
Total shipments, 1914			32,729,726	47,272,751	
Increase			14,543,025	19,385,715	
Per cent			44.3	41.0	

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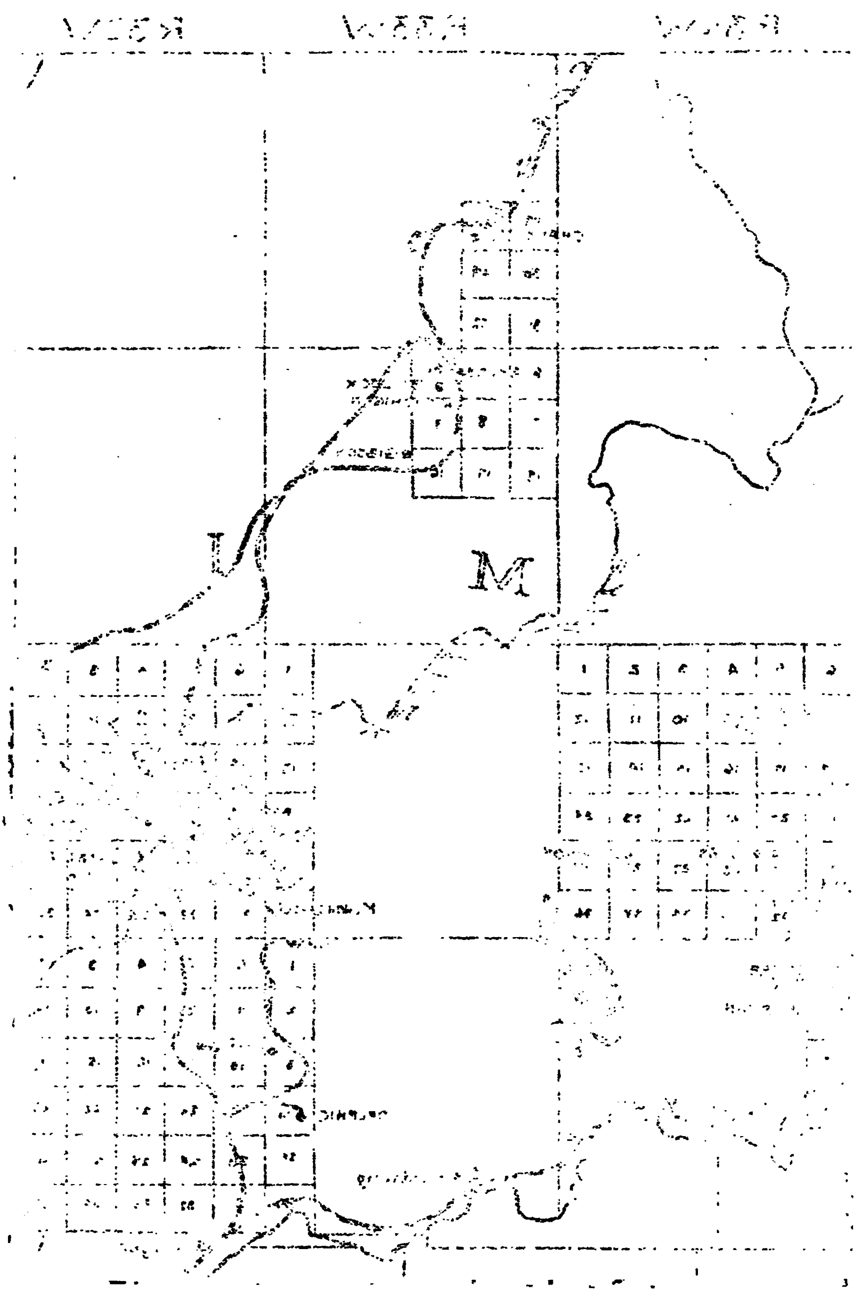
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